

AN INVESTIGATION INTO THE POSSIBILITIES OF

MECHANISED STOWING IN BRITISH COLLIERIES.



John Hunter Paterson,  
M.Sc., M.I.Min.E.

## P R E F A C E

This thesis records the investigation into the possibilities of the mechanisation of underground stowage of the goaf, with special reference to mining conditions in this country, and is the continuation of several years' practical research conducted in India on hydraulic systems of stowing sand.

Since the inception of this work three years ago, the subject has received increased attention from mining engineers. It is realised that the continued development of mechanised coal getting is dependent upon the successful mechanisation of the other operations in and around the coal face.

My special thanks are due to officials of the Production Dept., Scottish Division, National Coal Board, for co-operation given so readily at all levels, and to Professor R. Mc Adam who supervised the investigation.

Edinburgh.  
May, 1950.

J.H. PATERSON,  
M.Sc., M.I.Min.E.

## C O N T E N T S

### DEVELOPMENT IN MINING TECHNIQUE

Page.

GENERAL CONSIDERATIONS. . . . . 7.

### THE EFFECT OF COAL EXTRACTION ON STRATA

- (a) SUBSIDENCE:      (i) Introduction . . 12.  
                         (ii) Physical Aspects 13.  
                         (iii) Economic Aspects 16.
- (b) SYSTEMS OF STRATA CONTROL . . . . .
  - (i) The Caving System 18.
  - (ii) Strip Packing. . 18.
  - (iii) Solid Stowing. . 18.
- (c) CONTINUOUS MINING CONSIDERATIONS. . 20.

### GENERAL PREAMBLE ON THE NECESSITY FOR MECHANISED STOWING

- (a) INTRODUCTORY. . . . . 23.
- (b) FACTORS INFLUENCING THE INTRODUCTION  
OF POWER STOWING IN GREAT BRITAIN. 27.

### MINING LAYOUT FOR POWER STOWING

- (a) HORIZON SYSTEM OF MINING. . . . . 30.
- (b) METHODS OF WORKING. . . . . 32.

### STOWING MATERIALS

- (a) INTRODUCTION. . . . . 47.
- (b) NATURE OF STOWING MATERIAL: . . . . 49.
  - (i) Composition. . . 49.
  - (ii) Size Analysis. . 50.
  - (iii) Shape. . . . . 54.
- (c) COMPRESSIBILITY OF STOWING MATERIAL:
  - (i) Void Tests . . . 63.
  - (ii) Compression Tests 71.

## MATERIAL HANDLING

- (a) PREPARATION AND SUPPLY OF STOWING MATERIAL. . . . . 83.
- (b) TRANSPORT AND FEED ARRANGEMENTS:. . . . . 88.
  - (i) Pneumatic Stowing . . . . . 88.
  - (ii) Mechanical Stowing. . . . . 90.

## STOWING PIPES

- (a) INTRODUCTION. . . . . 94.
- (b) PIPE DESIGN:. . . . . 95.
  - (i) Material. . . . . 95.
  - (ii) Dimensions. . . . . 96.
  - (iii) Joints and Bends. . . . . 98.
- (c) WEAR IN PIPES:
  - (i) General . . . . . 102.
  - (ii) Flushing, Blasting and Impact Tests. . . . . 103.
  - (iii) Wear due to Friction and Impact. . . . . 107.
- (d) PRACTICAL PERFORMANCE:
  - (i) Sand-Water Mixtures . . . . . 110.
  - (ii) Air-Material Mixtures . . . . . 111.
  - (iii) Pipe Life and Tonnage Handled . . . . . 112.
- (e) CONCLUSIONS . . . . . 115.

## SURVEY OF STOWING PLANT

- (a) HYDRAULIC:
  - (i) History and Development 117.
  - (ii) Modern Plant & Practice 119.
- (b) PNEUMATIC:
  - (i) History and Development 123.
  - (ii) Modern Plant. . . . . 124.
  - (iii) Power Consumption . . . . . 129.
- (c) MECHANICAL:
  - (i) History and Development 131.
  - (ii) Modern Plant. . . . . 136.
  - (iii) Theoretical Considerations in New Designs. . . . . 139.



## DESIGN OF INBYE AIR COMPRESSOR

### (a) COMPRESSOR DESIGN:

(i) General . . . . .	144.
(ii) Blowing Pressure . . .	145.
(iii) Air Consumption . . .	145.
(iv) General Specification .	146.

### (b) DESIGN OF A SINGLE STAGE RECIPROCATING AIR COMPRESSOR FOR PNEUMATIC STOWING IN MINES:

(i) Practical Design Data:-	148
Capacity . . .	
Working Pressure	
Drive	
Cooling	
Dimensions	
Air temperatures	
(ii) General Considerations	148.
(iii) Speed	149.
(iv) Horse Power	149.
(v) Design of Electric Motor	150.
(vi) Piston Stroke and Piston Speed . . . . .	151.
(vii) Cylinder Diameter and Number of Cylinders .	151.
(viii) Arrangement of Cylinders	152.
(ix) Crank Arrangements . .	152.
(x) Cooling Water . . . . .	155.
(xi) Air Receiver . . . . .	158.
(xii) An Inlet Filter . . . .	158.

### (c) GENERAL DESCRIPTION OF PROPOSED COMPRESSOR . . . . . 160.

## THE ECONOMIC ASPECT OF POWER STOWING

(a) GENERAL ECONOMIC CONSIDERATIONS. .	164.
(b) LABOUR AND MANPOWER. . . . .	166.
(c) PLANT	
(i) Capital Cost. . . . .	173.
(ii) Depreciation. . . . .	173.
(d) POWER COSTS . . . . .	176.
(e) OVERALL STOWING COSTS . . . . .	177.
(f) CONCLUSIONS . . . . .	180.

## PRESENT APPLICATION AND FUTURE TRENDS IN POWER STOWING.

PRESENT APPLICATION AND FUTURE TRENDS IN POWER  
STOWING

(a) PRESENT APPLICATIONS . . . . . 184.

(b) FUTURE TRENDS . . . . . 187.

CONCLUSIONS . . . . . 189.

PRESENT APPLICATION AND FUTURE TRENDS IN POWER  
STOWING

(a) PRESENT APPLICATIONS . . . . . 184.

(b) FUTURE TRENDS . . . . . 187.

CONCLUSIONS . . . . . 189.

DEVELOPMENT IN MINING TECHNIQUEGENERAL CONSIDERATIONS

During the past thirty years there have been considerable changes in British mining practice, based on mechanisation of the coal getting operations, which resulted in a greater output of coal from a limited area, worked at a greater rate. The transporting of the coal from the face to the pit bottom has also followed well defined trends - face belts, gate belts and main trunk conveyors, and on to locomotive haulage.

As the introduction of machines has developed, so has the amount of dirt brought out of the pit increased, and it is estimated that at the present time there is from three to four times the amount of dirt being raised than was formerly the case. Whilst this is undesirable, with the system of mining now in vogue and the forced exploitation of seams of poorer quality, it must be considered inevitable, unless means are employed to stow or pack the excess dirt underground. The concentration of workings and the intense working of the faces has a significant effect on the subsidence question. On an average there are probably only about 50% of the faces working that there used to be, fewer roadways are maintained, generally in a better condition, which means a reduction in the total amount of packing required.

Under the above conditions it would be expected that the amount of subsidence would be greater but the available evidence tends to suggest that it is less. The probable explanation is that with more rapid advance of the face, the roof pressure is more evenly distributed and the closure of the goaf less complete.

The mechanisation of actual coal getting operations has been the focal point in the industry for many years and is again moving from its static position in the direction of cutter-loaders or other power loaders. We are fast approaching the position where any further advance in intensive working of the coal face will be brought to a halt by the non-mechanised, non-productive work. Very little has been done towards the mechanisation of this part of mining operations and it remains one of the most important problems facing the industry.

The question of power stowing is, however, now receiving increasing attention, its main development being in special circumstances where solid stowing was essential to the safe working of the colliery, or where the cost of strip packing was excessive. In the future the position will be that the packing and ripping will have to be brought in line with general face mechanisation to keep pace with continuous mining.



There is doubtless an extensive field for the application of mechanisation to reduce the amount of labour engaged on this work, but up to the present time there is no such machinery available for general use. There has, however, recently been a more general application of pneumatic stowing in this country, and some 25 installations are in use. Whilst the actual underground manipulation of this type of stowing is fairly simple, there are many other considerations which sometimes make successful application difficult and at times economically impossible.

The cost of crushing and transporting dirt from the surface to the workings is high, as also is the cost of producing the very large quantities of air needed for effective stowing. In this connection it may be stated that the high cost of pneumatic stowage is not the chief obstacle to its greater development. The practical difficulty of transporting the dirt back onto the coal face is more important. In modern mechanised mining an important feature for success is an efficient system for the transport of coal from the face. The more efficiently this is done, the more difficult it is to reverse the flow, as generally this system of one-way traffic is quite unsuitable for traffic in both directions.

The object of this investigation is to examine the possibilities of mechanised methods of stowing in British collieries, and to determine the most suitable methods in the light of prevailing conditions in the various coalfields areas, so that a practical and composite picture can be obtained as an aid to future research and development.

THE EFFECT OF COAL EXTRACTION ON STRATA

(a) SUBSIDENCE

- (i) Introduction.
- (ii) Physical Aspects.
- (iii) Economic Aspects.

(b) SYSTEMS OF STRATA CONTROL

- (i) The Caving System.
- (ii) Strip Packing.
- (iii) Solid Stowing.

(c) CONTINUOUS MINING CONSIDERATIONS

THE EFFECT OF COAL EXTRACTION ON STRATA(a) SUBSIDENCE(i) Introduction

The process of subsidence may be summarised by stating that when coal is extracted from an underground seam some degree of surface sinking follows. The amplitude of subsidence depends upon many factors, the most important being the depth and the thickness of the seam, the nature of the intervening strata, and the time factor in the face advance. It is now generally assumed that in the absence of special precautions the amount of subsidence to be expected is about two-thirds the thickness worked and that the area affected may extend beyond the actual workings a distance equal to approximately half the vertical depth of the seam being worked. *see page 13*

It has been proved both in this country and in other parts of the world, and particularly in the Ruhr, that whatever system of packing or stowing is employed, subsidence is bound to occur, because the loose material used is compressed by the weight of the superimposed strata. The primary use of packing or stowing is to support the roof, so as to control the movement of the undermined masses of rock with the object of maintaining the roadways and working faces in their proper dimensions and in a safe condition.

This may be achieved by roadside packs, strip packing or solid stowing. Generally there is insufficient debris on the spot for this solid packing, but when it is available the support should be better.

With hydraulic stowing large quantities of water have to be used to carry the stowing material through the pipes. This was found to be generally detrimental to the floor and working conditions, and has given way to pneumatic stowing, compressed air being used to force the debris into the waste. Alternatively, mechanical methods are being developed.

At Wemyss, using pneumatic stowing, 174,000 tons of coal were extracted from one panel and replaced by 139,000 tons washery refuse in addition to the stone retaining walls built from stone bands in the seam.

Surface Levels were taken before extraction commenced in 1939 and again in June, 1947, three years after completion, the maximum subsidence being 4' 3" over the centre of the panel. Allowing for further settlement, it can be assumed that the maximum total subsidence will be in the region of 5ft., approximately 25% the thickness of the seam and about half what would be expected without the use of pneumatic stowage.

#### (ii) Physical Aspects

The effect of the method of working and packing methods on the rate and amount of subsidence

? seam 20' thick  
2/3 of 20 - near 14'



is extremely important. The expression of a minimum time for the major proportion of the subsidence to take place and for complete stability to be reached is a matter for a great deal of consideration. The periods given by different authorities vary widely and it is obvious that geological considerations are again extremely important. Average conditions have been quoted in this country where total extraction has taken place, to be from two to five years. It is generally agreed that the maximum subsidence takes place in the first year after extraction, but complex geological conditions may delay or advance the initial rate of subsidence to a great extent. The amount of subsidence is usually expressed in terms of the ratio of maximum subsidence to the total thickness extracted. Obviously from considerations of the theories described it is impossible for the surface to be lowered to the full extent of the thickness of coal extracted, the maximum subsidence being in the region of 70 per cent. of the total seam thickness extracted. If the method of extraction is longwall and the system of working adopts various forms of packing of the goaf, in preference to complete caving, in which the roof is allowed to subside and close itself due to the downward load of the superincumbent strata, varying degrees of convergence of the roof to the floor will result. The effect of this variation in roof convergence will automatically result in a greater or less degree of

maximum subsidence of the surface. Experience in the Ruhr has indicated the following degrees of convergence expressed as a percentage of the same height:-

	<u>Per Cent.</u>
Full caving . . . . .	95
Strip caving. . . . .	23-90
Dummy road packing. . . . .	20
Hand packing (normal) . . . . .	60
Pneumatic stowage . . . . .	40-50
Mechanical stowage (throwing belts, etc). . . . .	40-50
Hand packing (carefully packed) . . . . .	40-50
Hydraulic stowage . . . . .	10-30

Similarly with regard to the time factor, experience on the Ruhr can be illustrated in the following example in which the time interval at which certain percentages of the possible maximum subsidence are related:-

<u>Period after working.</u>	<u>Percentage of total subsidence.</u>
1st year . . . . .	45
2nd year . . . . .	30
3rd year . . . . .	15
4th year . . . . .	5
5th year . . . . .	2
6th year . . . . .	2
7th year . . . . .	1

This illustrates that in general 90% of the subsidence has taken place after three years and 97% after five years. Thus if the seam were 5 ft. thick and the total possible subsidence 70%, or 4.2", after three years the surface would still subside 4.2", and after 5 years 1.26". Experiences in this country have suggested that the values quoted for rate and amount of subsidence can be generally applied to British conditions. It should again be emphasised, however, that there may still be cases where a longer period of settlement occurs and that geological knowledge/

knowledge is essential in order to obtain any degree of accuracy in such predictions.

There are large areas of coal lying under built-up regions in this country which in the national interest should be worked. It is conceivable that these coals can be worked with a minimum amount of damage to surface buildings, provided sufficient care is given to the planning of workings and the method of packing. A careful study of Levels during the early working of a seam can give the Mining Engineer vital information as to the amount of subsidence which may be reasonably expected from the method adopted.

### (iii) Economic Aspects

The cost of damage, throughout the country, caused by mining subsidence is very difficult to estimate. Such costs include the repairs to surface property, remedial and preventative measures, and purchase of support.

The National Coal Board gave evidence as to the estimated costs of surface damage for which the coal industry was liable under the existing law from 1936 to 1946 inclusive.\* These costs were based on returns for approximately 97% of the industry, adjusted to cover 100% of the production of coal in Great Britain. This cost ranged from £276,000 in 1936 (or 0.3d. per ton) to £511,000 in 1946, representing

---

\* Report of the Committee on Mining Subsidence.  
Cmd 7637, Mar. 1949.

approximately  $\frac{3}{4}$ d. per ton of saleable coal. The cost to the National Coal Board in 1947 covering 98.9% of the production was £611,000. These figures cover compensation to all types of surface property, including railways, the rising cost in recent years being explained by the increasing costs of labour and materials.

The Board estimate that in the future, apart from general influences increasing these costs, their liability under the existing law will be in the region of £1 million a year due to the following factors:-

1. The effect of the severance of coal and surface interests under the Coal Act, 1938, will be increasingly felt as undeveloped areas of coal are worked.

2. The practice of severing land without the right of compensation has ceased since Jan. 1939, except for land owned by the Board.

3. The increasing awareness of their rights by surface interests.

With the advent of the Town and Country Planning Acts, the question of damage to the surface and spoliating of amenities by surface tipping, etc., is likely to aggravate the position of the mining industry. The question then resolves itself into a matter of urgency to minimise the effect of subsidence by exploring every avenue of modern mining technique and mechanisation to the desired end.



(b) SYSTEMS OF STRATA CONTROL

The method of controlling the roof, especially in the goaf, has a significant effect upon road way and face conditions. Briefly there are three main systems of control:

(i) The Caving System - In this system the roof in the goaf is allowed to fall, and only roadside packs are built. It is economical in labour as it entails a relatively small amount of dead work. Rigid steel props or chocks are used to give a straight line of strong resistance at the goaf edge and so induce a shearing action. This system is suitable where good working conditions can be secured without detriment to surface subsidence, and the working of other seams. ✓

(ii) Strip Packing - Support of the goaf by strip packs is one of the most commonly practised systems in Britain. These packs built at right angles to the face vary in width according to local conditions the better the pack the wider the waste width between packs.

(iii) Solid Stowing - This system involves the complete filling of the goaf with packing material and usually gives improved conditions throughout the workings. It is specially used in seams liable to spontaneous combustion and where any considerable subsidence would affect adjacent seams or valuable surface rights. In this country it is practised only to a limited extent.



In all forms of packs the effectiveness of the support depends upon the materials used, the efficient building of the packs and the promptness with which they are extended as the face advances.

In addition to the support in the goaf, the face supports prevent falls of roof and coal and also give the earliest resistance to convergence. The use of steel supports has increased considerably during the past few years. Steel has a higher and less variable strength than timber, and in spite of the high initial cost its use is generally economic. These props take the usual form of H-section girders, or of drawn steel tubes, and are used in conjunction with wooden lids or bars and corrugated steel straps.

(c) CONTINUOUS MINING CONSIDERATIONS

Under the heading "Systems of Strata Control" a brief resume has been given of the present methods of roof control and face support, generally used in conventional systems of mining, where hand loading is still employed. Continuous mining involving the use of power loading demands that extra attention should be paid to a number of special factors. No matter how efficient the machine is, power loading cannot be applied successfully if roof conditions or timbering arrangements interrupt the continuity of loading. The use of these modern machines may mean the exposing of a large area of roof during actual loading operations, and it has been necessary to design special types of support such as screwtop props, quick release props and chocks, and forepoling. For the <sup>s</sup> faster, continuous moving faces of the future, it will be necessary to bring the face support position up to a mechanised standard and steps are now being taken to design a self-propelling chock. Such a "walking" chock would have to give continuous support whilst in motion. Similarly, face timbering must be designed to be rapidly erected and released.

The future of continuous mining is of such importance that every effort must be made to produce suitable conditions in which the process can be effectively worked. Mechanised stowing either of strip packs or of a solid stowed goaf is, therefore, a

corollary of continuous mining, more especially where the use of these machines is likely to be handicapped by bad roof conditions. Where main gates have to be ripped or brushed to keep up with an advancing face, the disposal of the debris calls for mechanised handling, either direct into the gate packs or into tubs for subsequent handling by mechanised stowing.

Rapidly moving workings will also have an important bearing on the question of subsidence. The speed of advance by these new working methods will lessen the distortion of the surface and will enable large areas of coal to be worked out and the surface let down evenly with little or no damage to surface structures. From an economical point of view alone, bearing in mind the subsidence damage figures quoted in the preceding sub-section, the possibilities and advantages of modern mechanised mining are tremendous and fully warrant the closest and urgent attention of responsible mining engineers.

GENERAL PREAMBLE ON THE NECESSITY FOR MECHANISED  
STOWING

(a) INTRODUCTORY

(b) FACTORS INFLUENCING THE INTRODUCTION OF  
POWER STOWING IN GREAT BRITAIN

GENERAL PREAMBLE ON THE NECESSITY FOR MECHANISED STOWING.

(a) INTRODUCTORY

One of the most important mining problems confronting the Mining Engineer is that of mechanised stowage of the goaf. Diminishing reserves of high-class coal do not permit the leaving of pillars of coal for support as was formerly the practice. The complete extraction of the available coal is an essential to the country's industrial future.

The system of controlling the roof in the goaf has a significant effect upon face and roadway conditions and upon the number of men occupied on deadwork.

A large percentage of underground workers are solely engaged on packing, especially in the more highly mechanised mines. Statistics collected covering persons so employed show that as high as 25% of underground personnel may be engaged on this work, and may, in solid hand stowing, rise to as high as one packer for every coalgetter.

Such heavy expenditure of labour brings home the necessity for the adoption of mechanised methods, but in the past little has been done in British mines in the mechanisation of this work. There is now, however, an awakening interest, and the Reid report recommended that the whole question of mechanised stowing was in need of investigation on a large scale, and that co-ordinated research with financial assist-



ance from the industry should be undertaken on both systems and appliances.

Another important feature is in a psychological sense. Packing and handling and shovelling stone debris is laborious and unpopular work and it is seldom possible to get men interested in the job; indeed, unless there is strict discipline and keen supervision, the work is badly and improperly carried out, with adverse effects upon roadway maintenance and roof control.

In general, by improved packing by mechanised means, there is the prospect of obtaining much better and safer roads than hitherto at a considerably less cost in labour and material both on the formation and later in maintenance which will have a bearing also on the efficiency and performance of all forms of coal transport.

In thick seams the difficulty of complete extraction, due to unsupported goaf means that a large percentage of coals is lost, and if the coal is liable to spontaneous combustion, dangerous conditions are created.

If all coal could be extracted and the goaf solid stowed, especially at rib sides, the elimination of underground fires would be complete.

Economical and safe working in the extraction of stoops in thick coal under conditions of limited subsidence or under sea workings demands solid stowing. Indeed, in practice, solid hydraulic stowing

has been successfully applied in several areas, and it appears that whatever method of solid stowing is used, there seems to be no other way by which stoops in thick seams liable to spontaneous combustion could be taken out with a high percentage of extraction and with the same degree of safety.

The need, in general, throughout the coalfields of Great Britain, for an economical system of mechanised stowing has long been felt. Many attempts with different plant have been tried in the past with varying results, the main object being to improve, by complete or partial stowing, the control of the roof at the face. This object has become more important with the planning and possible introduction of cutter loaders or other fully mechanised coal-cutting operations. Any system of stowing which will give maximum roof control will widen the scope for power loading.

Many methods of stowing have been devised by practical mining engineers in an attempt to contribute to a more systematic method of working with the attendant advantages in the safe and efficient working of minerals. Whilst, like so many other phases in mining, it is not possible to lay down any hard and fast rule as to the best procedure, there are certain common factors which can be investigated, and definite conclusions drawn. The final decision on any particular method is then mainly based on local conditions.

The usual practice of dealing with the void

from where the coal has been extracted by longwall working is by a system of hand built packs, but sometimes the caving system is adopted in which no packs are built except at the roadway sides. These systems provide a kind of roof control, but also unfortunately allow a general destruction of it, and marked convergence at the coal face which increases to an amount out by up to 60% until final settlement of the roadways has taken place.

It should be appreciated that the undercutting of coal by machine and the rate of advance with existing packing methods cause much greater destruction and faster lowering of the roof than in slower speed hand mining.

The common criticism against the introduction of solid stowing is that it is costly to install and operate, but it must be remembered, especially in machine mining, that the ordinary system of hand packing and withdrawing goafs is becoming a more and more costly operation with increasing losses of expensive materials and of coal left behind. These considerations, together with other far-reaching benefits that accrue from efficient solid stowing, leave no doubt as to its justification, and with future trends in power loading, is probably an essential in most seams to allow the fullest success of this new system of mining.

Of all the various operations in a mine, goaf stowing has resisted mechanisation the longest. The

principal demands made by the Mining Engineer on mechanised stowing is that it should be capable of dealing with large quantities of stowing material and keep pace with mechanised getting and handling of coal. The main reasons for this tardiness are the lack of sufficient quantity of stowing material of right quality and the handling of the material inbye to the site.

(b) FACTORS INFLUENCING THE INTRODUCTION OF POWER STOWING IN GREAT BRITAIN

Developments in power stowing are now taking place in this country and the reasons for the introduction and possible extension can be briefly stated:-

(1) To facilitate the extraction of thick seams worked by room and pillar methods, or in a series of lifts by longwall.

(2) To minimise the incidence of fire in seams liable to spontaneous combustion.

(3) To reduce subsidence in the case of undersea workings and to minimise surface damage or damage to seams in close proximity.

(4) To improve working conditions where a bad or tender roof requires extensive or solid packing.

(5) To reduce ventilation leakage through the goaf and in special cases to reduce the likelihood of excessive gas emission.

(6) As an economical method of introducing stowing material where a sufficient quantity of suitable material is not available in the pit.



(7) To absorb pit rubbish, with reduction of surface disposal cost, especially where surface room is restricted, or where such disposal would conflict with local amenities under the new Town & Country Planning Act.

(8) Where the results of power loading or continuous mining are being restricted by the non-mechanised bye work.

(9) To reduce working costs where hand packing is unduly expensive.

There are, however, certain secondary advantages which are cumulative and whilst not immediately apparent, are nevertheless important factors from which benefit will accrue. Amongst these can be cited improved general conditions underground, a reduction in the number of face and roadway supports distorted, all of which may contribute to a considerable saving in the overall underground costs.

The above factors, together with the safer conditions of work, are the main reasons for the trend towards mechanised stowing.



MINING LAYOUT FOR POWER STOWING

(a) HORIZON SYSTEM OF MINING

(b) METHODS OF WORKING

(a) HORIZON SYSTEM OF WORKING

In recent years the fundamental difference in underground layout between British and Continental mines has been the arrangements for transport. In the more level conditions in this country it was possible to follow the seams and form coal haulage roads in the seam, the haulage being first by hand, then by ponies and later by ropes. In the early working this system was satisfactory, but as the colliery developed and the use of a succession of rope haulages became necessary, the original advantage was lost and for the most part the outputs were limited by the inefficient haulage system. The development of the horizon system on the continent was undoubtedly due to the steepness of the measures and the necessity for driving level roads, firstly by following the contours in the seam which were later shortened to follow a straight and level course through the strata.

The general scheme of horizon mining may be described as the driving of two sets of main levels 100 to 150 yards apart, as stone mines from the shaft, and set out to follow the average level course of the coal seams. The upper horizon is usually planned to tap the highest workable seams in the royalty. From these main levels cross-cuts are driven at right angles to divide the area into panels. These cross-cuts are also driven in pairs, one above the other, and as they are usually on the same line

as the full dip of the measures, they cut the seams as they are advanced. It will be realised that the inclination of the seam and the distance between the horizons determines the length of face to be developed. In the flatter seams, in order to avoid faces of excessive length, it may be necessary to drive intermediate cross-cuts or sub-levels, or to connect the cross-cuts by staple shafts, so placed as to give the desired length of face. A suitable length of face is selected, dependent upon the mining conditions, so as to give an economical single unit, which is desirable for the operation of power stowing and power loading. Locomotives are used on the levels and provide an economical haulage system for both coal and materials.

The level at the low end of the face is used as the loader gate and can be equipped with belts, which can load directly on to the lower cross-cut or feed down spiral chutes in a staple shaft to a loading point in solid strata. The top level is used as the supply gate and stowing material and supplies can be taken in to the face efficiently, as the level is free from the congestion of coal transport.

It is apparent that this system of working possesses many haulage advantages over the common British practice, both as regards the supply and handling of stowage material and the transport of coal. Where extensive power stowing operations are contemplated in a new project, the provision of such an arrangement is necessary for success. Indeed,

the success of power stowing on the Continent, judged by the low manpower employed and the great quantity of stowing material handled, is due to the thorough transport arrangements resulting from the gravitational flow peculiar to horizon mining methods.

The considerable age of many British mines and the long distances from the shafts to the working faces, increase the difficulties in altering the layouts to accommodate modern locomotive haulage systems and from an economical point of view may be so formidable as to preclude the general large-scale introduction of power stowing in many cases. ✓

(b) METHODS OF WORKING

The three systems of power stowing packing material into the goaf can be applied to any system of mining, with the proviso in the case of hydraulic stowing that the seam conditions as to gradient are suitable. The actual face arrangements have become standardised through experience, with modifications to suit local conditions. Many different types of stowing support have been tried, including hessian cloth, wire lacing or wood struts. Where corrugated sheet can be used it is preferable from many points of view, as it allows a tighter pack to be formed and can be utilised again. This type of support is popular on longwall faces and Figs. 1 & 2 show a face boxed by this method. The writer devised and introduced into Indian mining practice another method which was found very satisfactory, <sup>and</sup> consisted of setting the



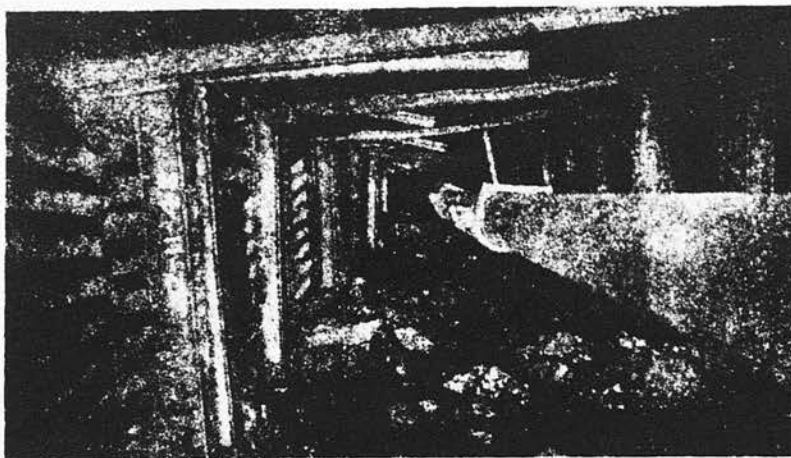


FIG. 1.—STEEL-SHEET STOWING SUPPORT BEFORE EXTRACTION OF SHEETS.

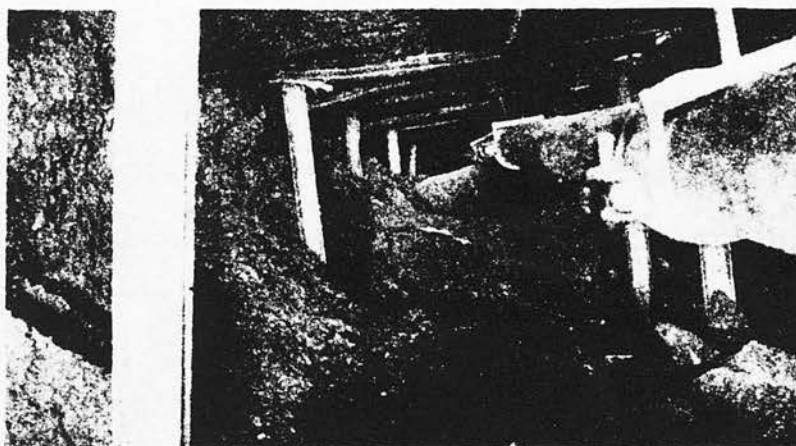


FIG. 2.—STEEL-SHEET STOWING SUPPORT AFTER EXTRACTION OF SHEETS.

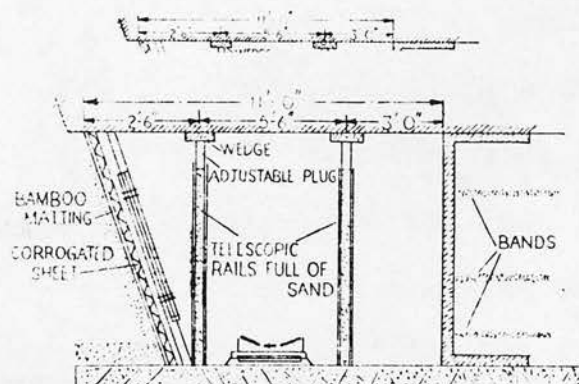


FIG. 3.—SECTION OF LONGWALL FACE READY FOR FIRST CUT.



stowing support at approximately the angle of repose of the stowed material. This allowed easy withdrawal of the support and a considerable saving in sheet loss. A sketch of this arrangement is shown in Fig. 3 for sand stowing. Normally the boxing support for room and pillar or bord and pillar is much more easily arranged and set, as by strip working a small coal partition is left to retain the packing. In certain instances it is possible to do without any stowing support, as where the roof is good the material can be allowed to take its angle of repose.

To illustrate some standard methods of working in conjunction with power stowing, brief descriptions of the layouts, using hydraulic, pneumatic and high-speed belt stowing are given.

Longwall advancing in conjunction with hydraulic stowing (India) - Fig. 4.

This layout shows a typical example of advancing longwall in a thick seam, as originally planned and developed by myself. The coal is cut and loaded on to a 26" face belt, the turn over being 5'6". The stowed pack is put on every two cuts, i.e. 11'0". The dip of the seam is 1 in 8 and the face line advances to the rise at 1 in 15, allowing ample gradient for water drainage away from the face. The sand pipes are brought in at the top end and will be crossed under the conveyor and down the face. The excess water from the face is led away to a settling sump to be pumped back to the surface.

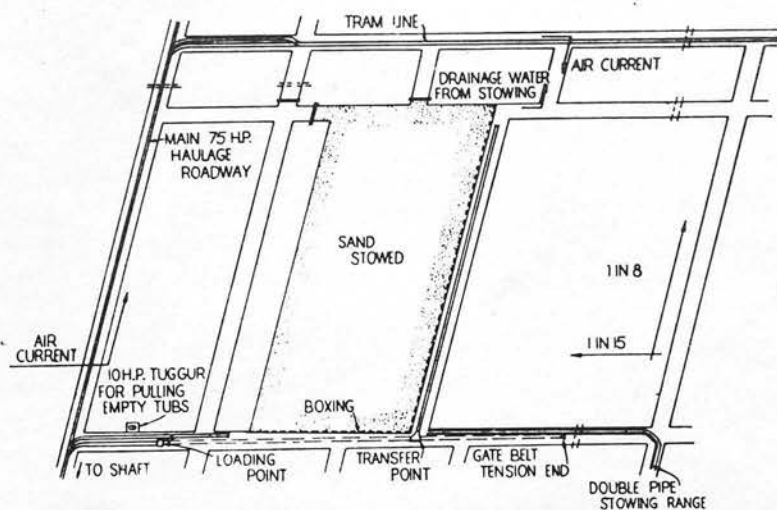


FIG. 4.—LAYOUT OF SAND-STOWED AREA, SHOWING LOADING AND TRANSFER POINTS AND GENERAL HAULAGE SYSTEM.

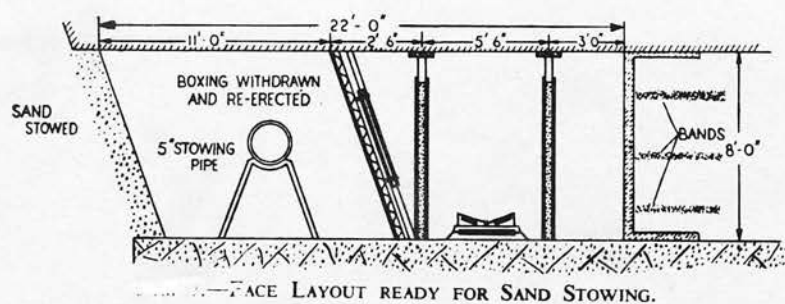


FIG. 5.—FACE LAYOUT READY FOR SAND STOWING.

The Latest Practice of Hydraulic Stowing in Poland.\*

In Poland seams of a thickness greater than 10 ft. are considered as thick seams. In order to avoid losses in coal recovery, to secure a higher degree of safety in working, and to avoid damage to the surface, regulations demand the application of back filling with sand in those seams. It is estimated that 50% of the total Polish coal output is obtained from seams thicker than 10 ft. This feature of Polish coal mining caused already in 1901 an introduction of systems of working with complete or partial back filling with hydraulic sand stowage. It is stated that the compressibility of sand stowing with fine sand is only 10% of the height. In thicker seams mined in layers, this percentage is even lower, which gives a greater possibility of coal recovery through partial or complete extraction under protected objects on the surface. In Poland some 14% of the total coal reserves is tied in standing pillars, so it can be appreciated the enormous importance of this system in securing the possibility of working this coal without leaving fenders.

In one case it was possible to work 50% of the coal under a city without causing any surface damage. By a full realisation of the planned extent of sand stowage, about 20% of the total reserves in Poland will be opened to mining. This means 15 billion

\* United Nations Economic & Social Council, 1949.  
Underground Mining and Coal Conservation.

tons of coal to a depth of 1,000 metres. To-day, at a yearly output of 70 million tons of coal, 11 million cubic metres of sand are being brought to the mines. At the time of the highest percentage of thick coal working in the total output, the quantity of sand will increase to 27 million tons per year starting in 1955.

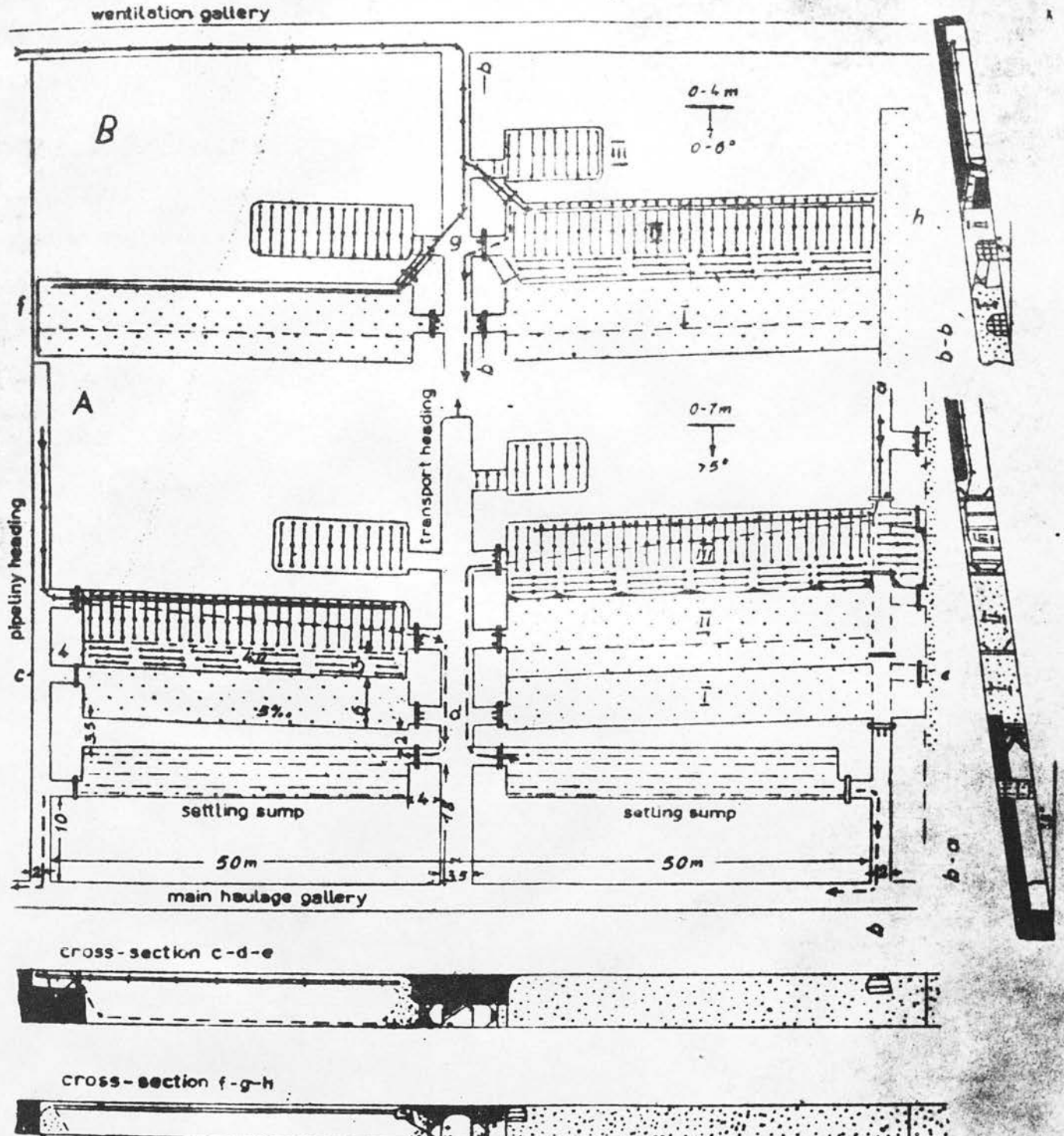
In conjunction with Hydraulic Stowing, the following three systems are in general practice.

System II(a) (Fig. 5) - This system consists of longitudinal room and pillar with the face moving upwards. From each butt entry the coal is mined in single or double panels, the panels being cut by horizontal galleries at distances of 50 metres. Beginning from the lowest gallery, a 6 metres wide room is driven up leaving a 3 m. wide fender against the previous working. The room is not cut through to the upper gallery at full width, but only at 2 m., where a wooden partition is built and through which the stowing pipes are introduced. The remaining pillar around the partition is taken later from the upper part.

For the drainage of water a stowing partition is also built in the lower gallery. From the latter a small wooden trough is led up to the upper dam and covered with short plates as the stowing advances. The trough serves as a water drain. In order to make possible the simultaneous driving of two rooms from one gallery, they are started as narrow openings 4 m. long.



# Longitudinal room and pillar system with hydraulic stowage (rooms driven on strike)



— FIG. 5. —



The stowing pipe line is installed only in the upper gallery. The drainage water after leaving the wooden trough is carried in steel pipes or covered wooden troughs to the settling sump in the main gallery, whence it flows to the main pumping system.

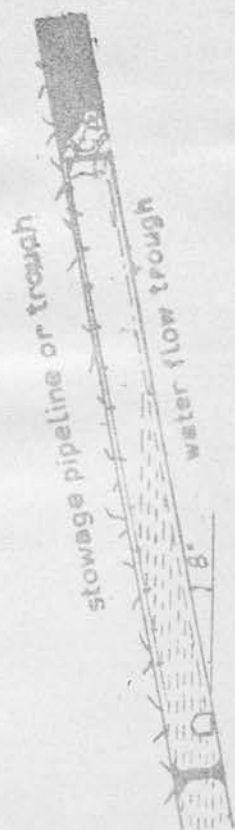
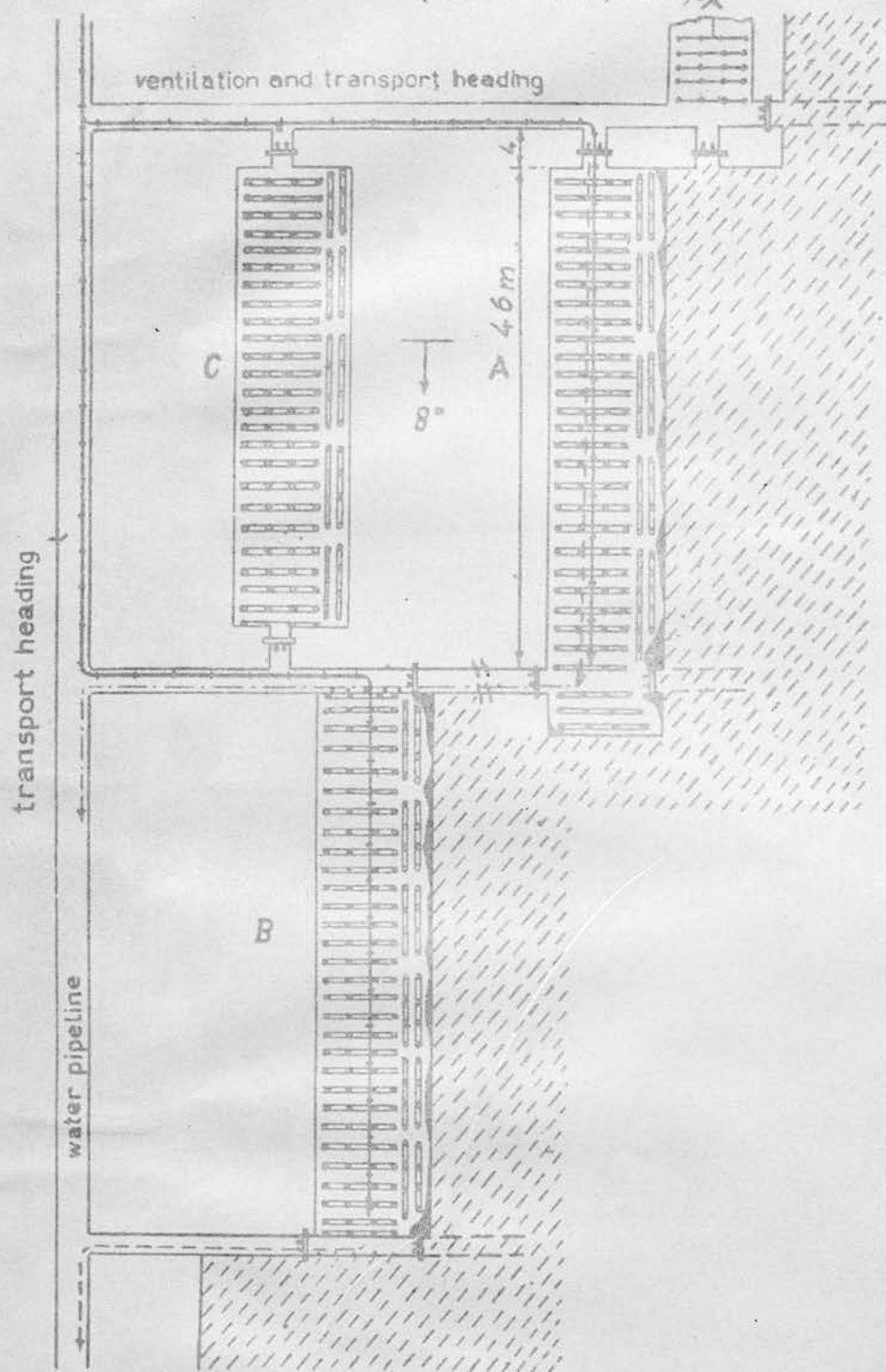
System II(b) - In less inclined seams mined in lifts, another system is applied as shown in Fig. 6, which allows a reduction in development work to transport and stowing pipe line headings. From the transport heading on both sides two narrow rooms are started which then broaden to 6 m., leaving a 3 m. pillar from the old workings. The room joins to the upper gallery with a small opening sufficient for the pipes. The stowing pipe line connected with the stowing shaft carries the sand/water mixture along an incline. The drainage water is carried to a clarifying sump above the main gallery.

The pillar along the pipe line heading is mined by driving rooms towards the sand-stowed part. The upper stowing partition is then set above the room in the heading. The heading is usually driven under the rock roof or in the higher layer. At a steeper dip the transport headings are used for the pipe line.

System III - The shortwall system with sand-stowing is shown in Fig. 7. It is similar in development as described in System **II(a)**. The coal face moves from the bottom upwards. The stowing partition is set up along the coal face at a distance of at least 2.5 m. and a side partition along the transport

*with a clear  
Description*

Longitudinal room and pillar system  
with hydraulic stowage  
(rooms driven upwards)

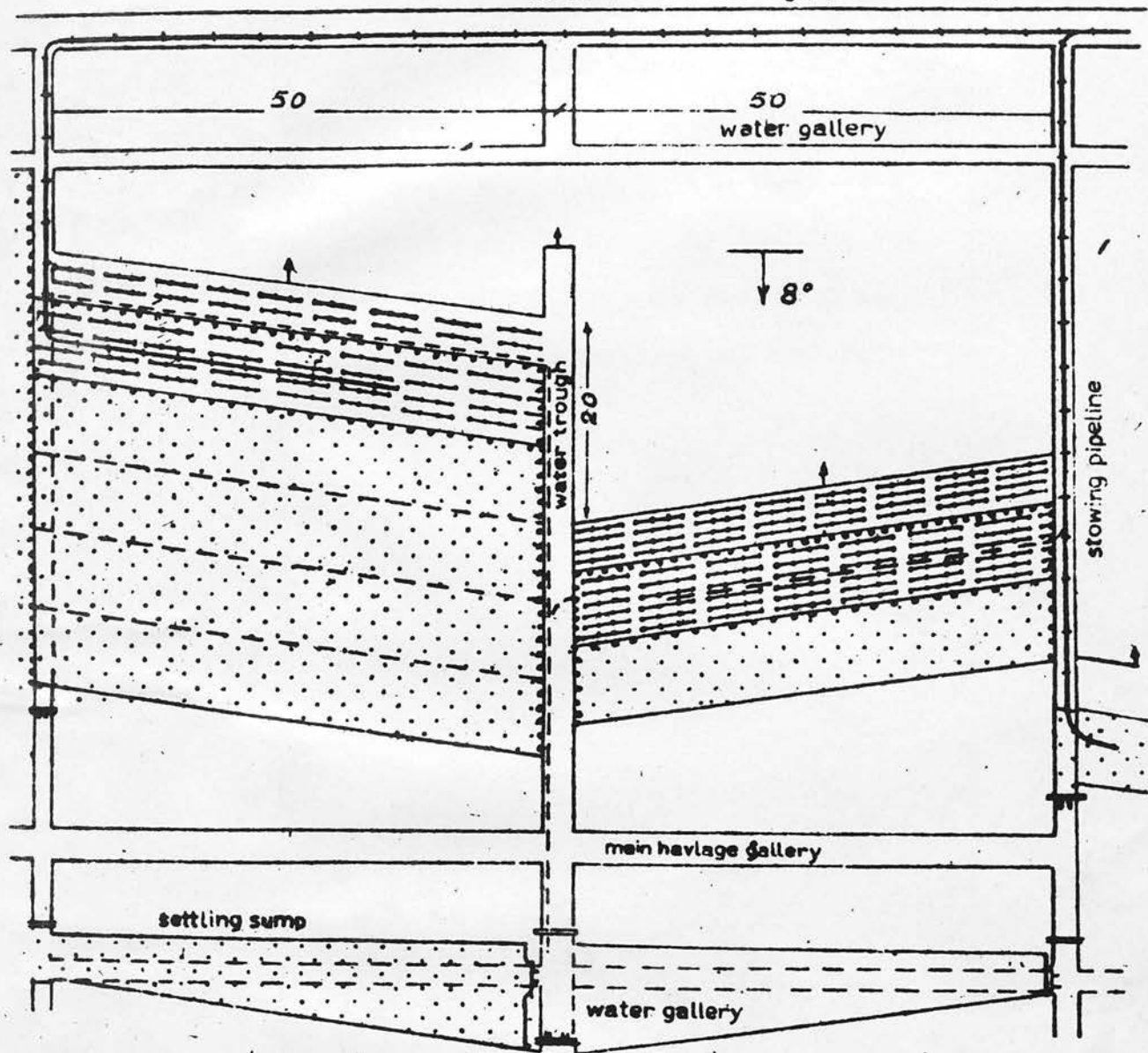


- Fig. 6 -

Fig. 16

## Shortwall system with hydraulic stowage

ventilation and stowing pipeline heading



- FIG. 7 -

heading. The stowage water is carried below the main pumping system. This system gives a clean recovery of coal, good stowing and continuous coal getting. To reduce the cost of wooden partitions, the surface of which is seven times larger than the room system, strong deal board brattices are used, which are moved forward with the advance of the face. The upper sections of the seam are developed independently from the lower ones. The coal face advances on the sand on the floor. By washing out sand at the side and on the floor, an undercut is obtained which results in less explosives and an increase in large coal.

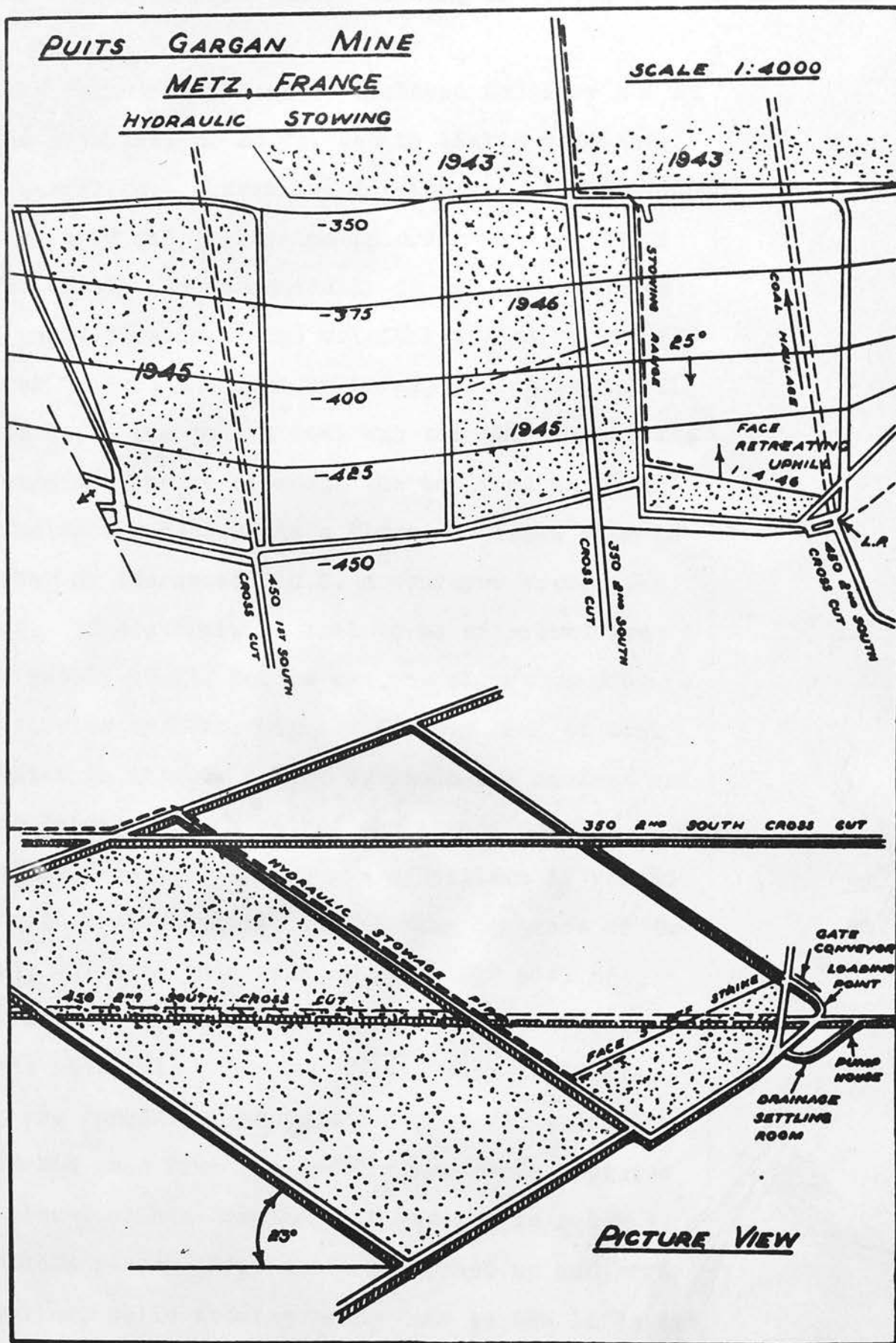


Longwall retreating in a Horizon Layout in conjunction with Hydraulic Stowing (Fig. 8). \*

A typical standard layout for this type of mining is the system adopted at Puits Gargan Mine, Metz, France. The seam 7'10" thick lies at a gradient of  $25^{\circ}$ . The longwall retreating faces are 300 ft. long and are offset  $5^{\circ}$  from strike which, together with the face retreating uphill, facilitates water drainage to the sumps. The settling sump and pump house are placed at the lowest part of the panel and connected direct with a crosscut, along which the return water is pumped. The stowing pipes come into the district along the upper crosscut and down the return airway to the face. The face is supported on wooden props and bars, each web of packing being formed of hessian cloth reinforced with wire netting. The timber was stowed in and not recovered. In these circumstances the pack was good and tight to the roof. In this case a convenient sand quarry on the surface supplied the material, which also included ashes from the colliery boilers.

\* B.I.O.S. Report No. 1380.

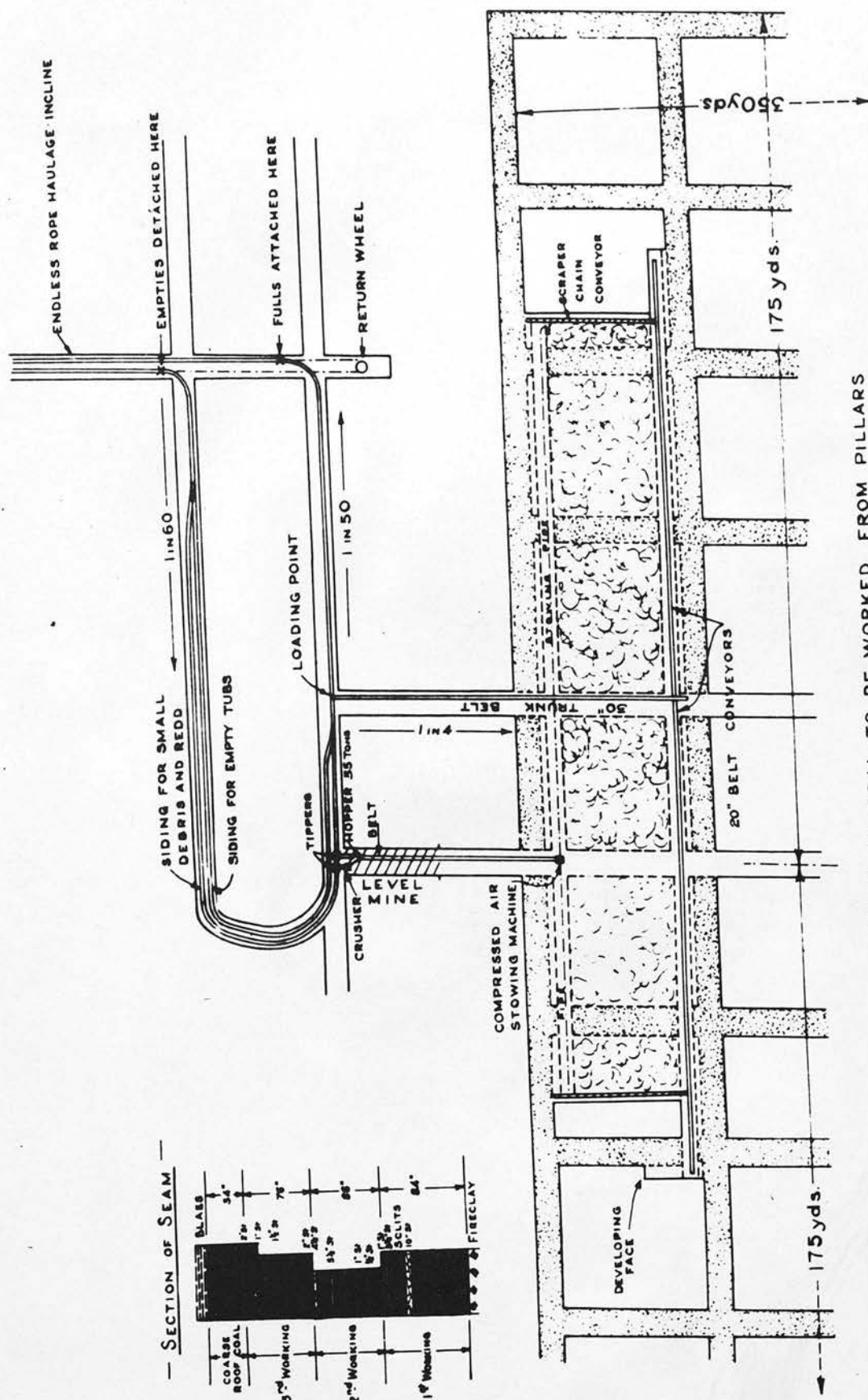




- Fig. 8. -

Stoop Extraction in Thick Coal (Dysart Main)  
in conjunction with Pneumatic Stowing (Fig. 9).

The Dysart Main Seam at Lochhead Colliery has an average thickness of 20 ft. and is liable to spontaneous combustion. Extensive development had taken place by bord and pillar and in order to work these pillars safely and economically it was necessary to adopt solid stowing. The original method of extraction was to work the coal in lifts, taking the middle 6ft. first. The bottom coal was then worked and the excavated area solid stowed; the top coal was then taken using the packing as a floor. With a view to obtaining an increased O.M.S. a conveyor system was adopted. The pillars of coal to be extracted are approximately 90 ft. square and the old rooms measure 15-20ft. wide by 13ft. high. The top leaf of coal 6ft. thick is intact. Fig. 9. shows the haulage and stowing layout and the method of extraction of a bord and pillar panel. The top row of pillars is worked out first to the limit of the blowing distance of the machine, which in this case is about 175 yds. on either side - this distance being variable depending upon the material. The two faces, each 25-30 yds. long, are formed on the bottom leaf 7 ft. thick and are worked away from the trunk conveyor in opposite directions. When the limit of stowing is reached, the middle portion 7'0" thick is opened up and worked retreating, solid stowing being done as the lifts are cut. The adjacent row of pillars is extracted next in a similar manner. The top leaf 6ft. thick not



AMOUNT OF COAL TO BE WORKED FROM PILLARS  
IN THIS PANEL = 625,000 TONS.

Fig. 9. - Lochhead Colliery. Showing method of stoop extraction in  
Dysart Main Coal.

being cut in the rooms, is solid throughout and a much longer face will be worked subsequently on top of the solid packing filling the first and second workings. The top coal will only be worked when a complete panel is ready for extraction.

Longwall advancing in Thick Coal (Dysart Main) in conjunction with Pneumatic Stowing (Fig.10).

The district being worked at Michael Colliery is in an area of undersea coal in which the seam is 28ft. thick including stone bands. The plan shown in Fig.10. gives the haulage layout and stowing arrangements. It will be seen that the panel was originally laid out for faces of 200 yds, but this has been abandoned for one approximately half this length. The present face in the bottom leaf is 110 yds. long and is taking a 5ft. lift working westwards. The coal is undercut 4ft., the turnover being on a 48 hour cycle and produces 110 tons. It is proposed to increase the undercut to 5ft. and have a daily turnover to obtain 275 tons per shift.

When the first working face reaches the limit of the panel, another working face will be opened up at the outbye end, taking the second lift of 5ft. where the first working commenced. This also will be worked to the panel limit with solid stowing. The same procedure will be followed in the third working, but the fourth lift may be worked retreating.

The stowing machine is fed direct from a hopper pit situated conveniently at the top of the panel, so that the whole panel can be stowed without moving the



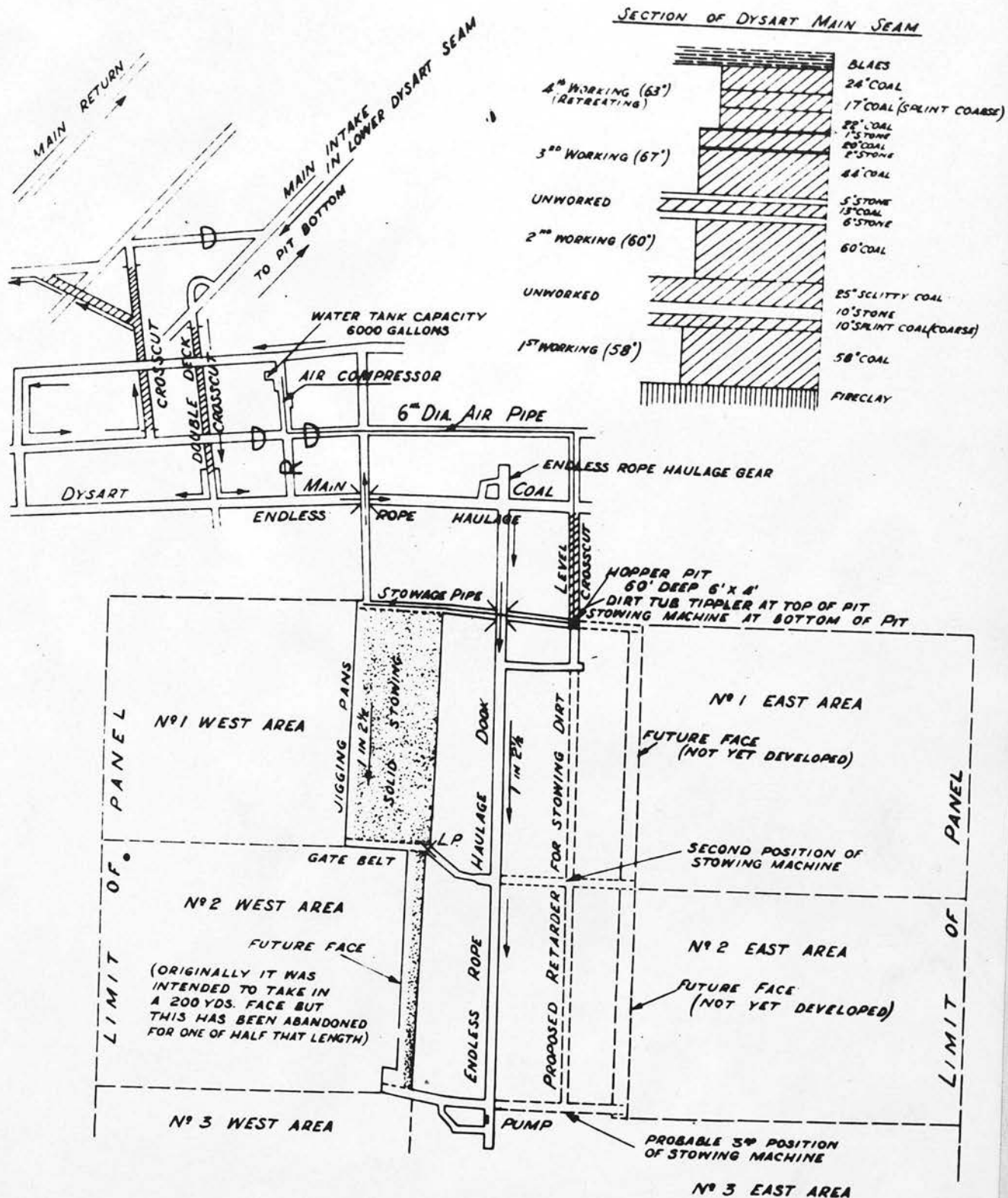


Fig. 10. - Michael Colliery. Method of working Dysart Main Coal.  
Waste stowed solid by the pneumatic method. Scale:  $\frac{1}{2}" = 66'$

machine other than turning it to face the opposite direction, for stowing the East panel. The output thus won will be in the region of 200,000 tons. The machine will then be moved down to the position shown on the plan and the East and West panels worked as before, the stowing dirt being fed from the hopper on to a retarding conveyor to the stowing machine.

Longwall advancing and strip packing by means of pneumatic stowing (Fig.11).

The Barnsley seam in South Yorkshire has always been particularly liable to spontaneous combustion. The heatings occurred mainly in ribside roadways, and in the layout of the faces special attention was directed to minimise the number of ribsides. At Bullcroft the seam lies at a depth of about 700 yds. and dips at an average gradient of 1 in 20. The seam section worked consists of 6'3", which includes a band of clod 8" to 1ft. thick. Immediately above this is the Connie coal which forms the roof and is partially recovered in the waste. There is another band of carbonaceous shale underlying another coal called the Daybeds, and this is an indirect contributing factor in the spontaneous heatings, together with the presence in the gate side packs of the clod or shale. A system of hand packing was introduced, using imported inert material, packed by specially selected teams of men on day wage. Whilst this was successful, it was very expensive in manpower and thus pneumatic stowing was introduced. Power stowing the gate side packs effectively stopped the heatings and no further trouble from this cause has been experienced. The improved conditions at the face allowed the widening out of the wastes to 12 yds. and the packs to 7 yds. as against 8 yds. waste and 3 yds. pack under the old system. The Bullcroft system proves the application of pneumatic stowing for strip





packs and gives an indication of the probable use of this type of mechanised stowing to keep pace with a fully mechanised face.

Longwall retreating in conjunction with high-speed belt stower. (Fig. 12) \*

Whilst at present high-speed belt stowing in this country is in the experimental stage, there are a number of such units at work in Germany. Fig. 12 shows the layout of the Albert I seam in the Friedrich Heinrich Mine. This shows a unit which was first worked by advancing longwall and then a face being worked retreating. It will be seen that the face lies almost level and the stowing dirt is brought down from the 350 metre level by means of a staple pit. The dirt is then taken by conveyor and thence into the machine. The stower used was the German Schleuder and the packing done from the tension end of the face conveyor towards the driving head, i.e. the opposite way to pneumatic stowing.

As is the general practice, faces are usually set out as single units advancing or retreating on the strike, with the loader gate at the lower end of the district.

\* Ruhr Coalfield Tech. Report BIOS, No. 1380.



From the few layouts described, it will be seen that in all different conditions and systems of mining, mechanised stowing can be adopted with advantage. Pneumatic stowing appears to be the easiest to instal and the most flexible in operation at the present time. From the two Continental layouts described it can be seen clearly that the horizon method of working possesses certain advantages as regards the supply and handling of stowing material.

The above operational advantages are not common to British practice and any introduction of mechanised stowing in this country presupposes the economical inbye handling of the material either using the present haulage layout or the possible alteration to more suitable means - locomotive or belt - depending on many local factors. The question of handling large quantities of stowing material especially in steep workings would mean the adoption of full or a partial system of horizon mining.



STOWING MATERIALS

(a) INTRODUCTION

(b) NATURE OF STOWING MATERIAL

- (i) Composition
- (ii) Size Analysis
- (iii) Shape

(c) COMPRESSIBILITY OF STOWING MATERIAL

- (i) Void Tests
- (ii) Compression Tests

STOWING MATERIALS(a) INTRODUCTION.

The nature of the stowing material is of paramount importance, as the amount of support given by the packs depends directly on the compressibility of the material used, together with the completeness of the packing. Consideration of this point is a first essential in regard to the choice of material, where more than one kind may be available. Other strength characteristics to be taken into account beside compressive strength are elasticity, toughness and particle shape, as these also influence the behaviour of the packs under pressure. The importance of the strength properties has not been fully investigated.

Available material can be conveniently divided into two classes as far as source of supply is concerned:-

- (1) Material available without preparation;
- (2) Material available after preparation.

The selection of a suitable size and type of material depending upon whether hydraulic, pneumatic or mechanical plant is to be used may necessitate admixtures of several classes of material in order to obtain the best results. The effect of water on the material is also important as it is advisable to damp all material (unless naturally damp) before entry into the pipes for the following reasons:-

- (1) Dust prevention
- (2) Lessens abrasion in pipes, etc.
- (3) Damp material forms a tighter pack.

In hydraulic stowing the material must not disintegrate to form sludge, as this will tend to cause trouble with blocked pipes. The amount of water to be added should be determined by trial and may vary from 2 - 7%. Correlation of all work done on this subject is necessary so that a standard test of material suitability can be formulated.

(b) NATURE OF STOWING MATERIAL

(i) Composition

Many types of material can be used for mechanised stowing, either alone or as mixtures:-

Crushed pit rock or shale, etc.  
Washery debris.  
Sand, gravel or earth.  
Blast furnace slag.  
Boiler ash or flue dust.

The first three types are most generally used, more from availability than suitability. Mixtures in varying proportions seem to be the most suitable. The ideal material should be of such a composition that does not disintegrate in transit nor become sticky with admixture of water. These two factors have an important bearing when estimating the probable length of conveyance in pipes and the quantity of conveying medium, i.e. air or water, required.

Heavy, highly abrasive materials and soft clayey materials require special consideration and in these cases the mixing of other materials in suitable proportions may prove of considerable benefit. On the Continent, heavy river sand has been mixed with gravel in the approximate ratio of 5 to 1, and sticky washery material with a similar proportion of sand or ashes to make a successful stowing material. In this country it would appear from enquiry into present materials used, that a mixture of pit rubbish, washery debris and crushed rock from underground rippings can be stowed with



success and where possible a small percentage of boiler ashes has been found to increase the suitability of the mixture. In general, coal measure strata provides a suitable stowing material, admixtures varying for each class of shale or rock, determined by practical test for correct proportion.

### (ii) Size Analysis

The sizing or grading of the packing material controls the density with which it can be packed by a given method of packing. For every material there is a certain arrangement of sizes which will give a mixture of maximum density or minimum percentage of voids. Any departure from this grading, such as an excess of fine material will result in a reduction in the density of the mixture.

The limit of sizes varies with the system of stowing, and as far as hydraulic and pneumatic methods are concerned, is closely linked with pipe diameter. In pneumatic stowing it is generally accepted that the pipe diameter should be twice the maximum round material size, i.e. a 4" diameter pipe for 2" material, 6" diameter pipe for 3" material, etc., etc. This latter size is a convenient size and appears to be suitable for general application. In the case of hydraulic stowing, the percentage of scouring size (above  $\frac{1}{2}$ " ) can be much lower - that is, small-sized material can be successfully handled without considering the larger sizes as is necessary in pneumatic stowing. On the other hand, the

material must not contain too great a proportion of the smallest sizes which may tend to form sludge which will be difficult to hold in the packs and thus cause considerable trouble in sumps. In mechanical stowing the limiting size appears at the moment to be about 4", but if necessary this can be increased with little difficulty. Typical size tests of samples of stowing material as used in Scotland are given in Table 1.

Size	Percentage by Weight		
	Mixed Table Pickings	Crushed Shale	Washer Redd
3" - 2"	13.3	11.7	17.4
2" - 1"	42.8	54.1	31.8
1" - $\frac{1}{2}$ "	13.5	20.0	26.3
$\frac{1}{2}$ " - $\frac{3}{4}$ "	10.1	8.3	15.7
$\frac{3}{4}$ " - $\frac{1}{8}$ "	9.6	3.5	6.3
$\frac{1}{8}$ "	10.7	2.4	2.5
	100.0	100.0	100.0

Tests have been made by other workers and for comparison are given below:

Size test of washery stowing dirt and crushed material as used in South Yorkshire.\* Table 2.

Size	Percentage by Weight	
	Washery Dirt	Crushed Material
3" - 2"	11.9	17.8
2" - 1"	35.4	36.8
1" - $\frac{1}{2}$ "	36.3	23.0
$\frac{1}{2}$ " - $\frac{1}{4}$ "	8.8	9.5
$\frac{1}{4}$ " - $\frac{1}{8}$ "	2.9	5.3
$\frac{1}{8}$ "	4.7	7.6
	100.0	100.0

\* Whetton & Sinha., Coll. Guard., 17.11.49.



Table 3.

Size test of stowing material in S. Wales.<sup>XX</sup>

Size	Percentage by weight		
	Fresh washery shale	Rubbish Tip	Unstowable Material
Above 1½"	14.3	4.8	1.6
1½" - 1"	9.2	14.8	2.2
1" - ½"	18.6	18.1	6.2
½" - ¼"	16.8	18.6	16.0
¼" - 1/8"	11.2	8.6	20.2
1/8" - 1/16"	9.1	9.5	17.3
Below 1/16"	20.8	25.6	36.5
Moisture	7.2	5.9	6.3

From the tests it can be deduced that a wide range of sized material can be successfully stowed by pneumatic means. It will be seen that washery dirt used in Scotland and Yorkshire has a very small percentage of minus 1/8th" which will give trouble free running in the pipes, especially as there is also a good quantity of the heavier or scouring size material in the mixture. It would appear from a close examination of numerous size analysis charts that the ideal material, to avoid chokes in the pipe and still give a good pack density, should contain not more than 25% of minus 1/8th material, if of a hard nature, and not more than 15% if of a friable nature, to allow for degradation in pipe travel. This with 50% above 1" would give an efficient packing medium. (Ref. void notes.)

<sup>XX</sup> Lansdown, Coll. Guard. 24.2.49.

This question of size stowability is closely allied to the moisture content, as excess moisture will tend to increase the range of settling material due to skin friction in the pipe and thus the percentage of scouring size becomes important to ensure efficient air conveying. It is obvious that with dry material the position is different and as far as conveying in pipes is concerned, the percentage of small material even of dust size is immaterial.



### (iii) Shape

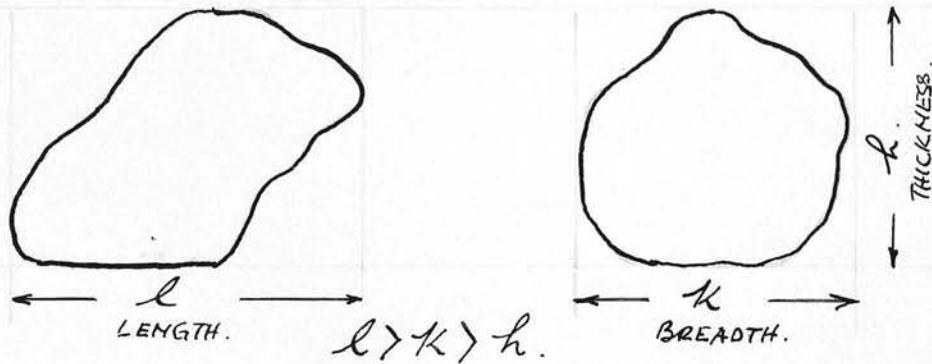
The shape of the pieces of stowing material governs the way they fit into the pack and have an important bearing on the subsequent load distribution in the pack. For example, pieces which have a pronounced cleavage and lie parallel with the joint planes to the floor may be expected to develop a high mechanical strength, whilst irregular shaped material having point contacts with each other when in a pack, tend to crush and break under relatively small loads. The efficiency of movement in a confined space under air or water pressure is directly dependent on the exposed area of the body being moved. The shape of each individual particle of material thus has an important bearing on the stowability of the material. Material which breaks down to spherical or cubic fractions is much more efficiently carried by an air/water stream than flat, thin pieces. A simple shape test of length to breadth and thickness will show whether the method of crushing is suitable for the material. Taking the ideal ratio as unity proportions, it can be generally accepted that the ratio of length to breadth of 1 to 3 and thickness/breadth up to 1 to 5 are good stowing materials.

In dealing with shape, the particle shape is determined by the percentage of flaky and elongated particles they contain. These are defined respectively as particles whose least dimension is less than 0.6 of their mean size and whose greatest

dimension is more than 1.8 times their mean size.

A good method of test is as follows:- The material is first sorted on a B.S. square aperture test sieve into a number of closely-limited particle size groups, i.e.  $1'' - \frac{3}{4}''$ ,  $\frac{3}{4}'' - \frac{1}{2}''$ , etc., and each group is tested for length and thickness on appropriate gauges. Flaky particles could be separated more rapidly on slotted sieves of the appropriate dimensions, but this would mean a special sieve for each particle size group, whereas one thickness gauge can be used for all. It is difficult to determine a good test for roundness except by direct measurement.

The Nature of Shape - Material of any given size will contain stone of many shapes, and the problem of determining the shape of the aggregate resolves itself into a statistical study of the geometry of the stones. Before considering this problem it is necessary to decide what is meant by the shape of any individual stone. In the first place there is the relative roundness or angularity of the material which it is convenient to term the form and in the second place there is the shape proper by which is meant the extent to which the particle may be said to be flaky or elongated.

FIG. 13.

The main practical factor involving shape is the effect of shape on the surface area. The efficiency with which a particle of stowing material will be carried along a pipe line by either compressed air or water is to some extent governed by the density and the surface area. The relation between surface area and weight of both spheres and cubes is given by the expression:  $S = 6/pa$  where  $S$  is the surface area in sq. cms. per gram,  $p$  is the density of the spheres or cubes and  $a$  is the diameter of the sphere or side of the cube in cm. This formula, however, does not take into account the particle shape, but allowance can be made by introducing a shape factor  $K$ , the expression then becomes  $S = 6K/pd$  where  $d$  is the mean sieve size. This mean sieve size is the arithmetic mean of the two consecutive B.S. sieves used to define the size of the material, e.g. if the material is  $1\frac{1}{2}$ " -  $\frac{3}{4}$ ", the mean size in inch units is  $7/8$ " (0.875). As the shape factor plays such an important part in the theory of conveying by air/water, experiments were made to

determine the factor K for two common types of coal measure rock used for stowing material, viz. shale and sandstone. The shape of crushed material of given sieve size depends only on length and thickness; since, however, both weight of a particle and its surface area increase very nearly directly with the length, it follows that the shape factor K will depend almost entirely on the thickness of the particle.

A sieve size of  $1\frac{1}{2}$ " was taken, as this covers a good percentage fraction of stowing material and sorted into thickness groups. The surface area was determined by carefully wrapping each stone with fine lead foil and the foil required to cover the stone was weighed. Then by weighing a known area of the foil the surface area of the sample was obtained by direct proportion.



Shape Constant - Representative samples of sandstone and shale stowing material lying between sieves 1" and  $\frac{1}{2}$ " were measured for length, breadth and thickness. The weight, volume and surface areas were also determined. The results are tabulated in Tables IV and V:-

Table IV

Material - Sandstone; size 1" -  $\frac{1}{2}$ ", mean size d  $\frac{3}{4}$ ", specific gravity 2.46.

No.	Length l cms.	Breadth K cms.	Thickness h cms.	Weight grms.	Volume c. cms.	Surface Area sq. cms.
1	5.50	2.40	2.45	35.05	14.2	46.15
2	4.70	2.35	2.35	28.90	11.74	38.46
3	3.15	2.50	2.55	23.45	9.38	24.61
4	4.35	2.25	2.40	19.70	7.90	27.69
5	3.80	2.45	1.80	15.15	6.06	26.92
6	3.40	2.30	1.65	14.80	5.92	23.07
7	3.35	2.55	1.20	12.25	4.92	20.00
8	2.4	2.25	1.65	9.35	3.74	16.92
9	3.75	1.70	1.45	9.15	3.65	15.38
10	2.30	1.85	1.05	5.85	2.34	12.30

TABLE V

Material - Shale; size  $1'' - \frac{1}{2}''$ , mean size  $d \frac{3}{4}''$ ,  
specific gravity 2.22

No.	length l cms.	breadth Kcms	thickness h cms.	weight grms.	volume c.cms	Surface Area sq.cms.
1	5.35	1.80	2.20	20.80	9.36	30.76
2	6.30	2.75	1.35	19.20	8.64	32.30
3	4.30	2.45	2.10	18.00	8.10	29.23
4	4.35	2.55	1.80	17.75	8.00	27.69
5	3.40	2.50	1.75	17.10	7.70	23.76
6	4.15	2.15	2.25	16.50	7.43	21.52
7	4.65	1.75	1.55	15.45	7.0	24.61
8	4.05	2.50	1.60	13.3	6.0	23.76
9	4.20	1.70	0.95	7.05	3.17	16.92
10	2.80	2.10	1.25	7.40	3.33	13.84

The surface area measurements were made as described previously. A sheet of lead foil 10 cm x 10 cm. was carefully weighed, the weight being 0.65 grammes. Therefore 1 gram of lead foil is equivalent to 153.8 sq.cm. and 1 sq.cm. weighs .0065 grms. By weighing the amount of foil required to cover the material, the equivalent surface area can be found by direct proportion.

As stated previously, by introducing a shape factor K

$$\text{Surface Area } S = \frac{6K}{pd} \text{ sq.cm./gm.}$$

$$\begin{aligned} \text{therefore } K &= \frac{Spd}{6} = \frac{S \times 1.90 \times 2.46}{6} \\ &= \frac{4.67S}{6} = .78S \end{aligned}$$

Values of K for sandstone calculated as above are shown in TABLE VI:-

Sample	Weight in Grammes	S	Surface Area	Shape Constant
1	35.05	1.31	46.15	1.02
2	28.90	1.33	38.46	1.04
3	23.45	1.05	24.61	.82
4	19.70	1.40	27.69	1.09
5	15.15	1.77	26.92	1.38
6	14.80	1.55	23.07	1.20
7	12.25	1.63	20.00	1.27
8	9.35	1.80	16.92	1.40
9	9.15	1.68	15.38	1.31
10	5.85	2.01	12.30	1.56
Average...				1.20

It would appear that for coal measure sandstone the shape constant varies between 1.0 and 1.5, an average of ten examples giving 1.20.

The values of K for shale are shown in TABLE VII.

$$\text{Surface Area cms./gramme} = \frac{6K}{pd} \quad p = \text{mean size}$$

$$K = \frac{Spd}{6} = \frac{S \times 1.90 \times 2.22}{6} = .703S$$

TABLE VII

Sample	Weight in Grammes	S	Surface Area	Shape Constant
1	20.80	1.47	30.76	1.030
2	19.20	1.67	32.30	1.174
3	18.00	1.62	29.23	1.139
4	17.75	1.56	27.69	1.096
5	17.10	1.38	23.76	0.970
6	16.50	1.30	21.52	0.914
7	15.45	1.50	24.61	1.054
8	13.30	1.78	23.76	1.251
9	7.05	2.40	16.92	1.680
10	7.40	1.87	13.84	1.509

Average 1.18

For shale the constant lies between 1.0 and 1.68, the average of ten examples giving 1.18

The results show that the constant K varies with the relative thickness of the material. For average material using the constant = 1.2 for sandstone and 1.18 for shale, the surface area for a 1" -  $\frac{1}{2}$ " fraction was reasonably correct. For these values

$S = \text{surface area/sq.cm./gm.}$

For sandstone = 7.2/pd

For shale = 7.08/pd

where d = mean size in cms.

p = sp.gravity of material.

These formulae give a convenient method for estimating the surface area of material lying between sieves when the specific gravity is known. The



shape of material in terms of surface area can be assessed and thus, its practical use when air borne or water borne. These factors will have some bearing on the crushing of rock to sizes suitable for conveying and to the method of crushing, to give suitably-sized material with maximum surface area, that is, to avoid flaky and elongated particles.

(c) COMPRESSIBILITY OF STOWING MATERIALS(i) Void Tests

The density or compactness of a pack depends upon several characteristics of the stowing material, viz; type of material, size, shape and degree of wetness, together with the force with which it is packed into the goaf. The combined effect of the above factors can be expressed in the percentage of voids contained in any pack. The direct accurate measurement of the hollowness of a pack underground is difficult, but laboratory tests on bulk density of the material to be used in its construction can give an indication of the probable resultant make-up of the pack. The bulk density of material in lbs. per cubic foot can be determined from the weight of compacted material contained in a standard measure of  $\frac{1}{2}$  or 1 cu. ft. capacity. The percentage of voids is determined from the bulk density (W.) and the specific gravity (S.G.) by the formula:

$$\text{Percentage of voids} = \frac{(\text{S.G.} \times 62.4) - W.}{\text{S.G.} \times 62.4} \times 100$$

If the specific gravity is not known, the voids may be conveniently measured by the amount of water required to fill them using a glass cylinder, 5" diam. and 12" high, graduated so that parts A and B are of equal volume. (See sketch.) Fig. 14.

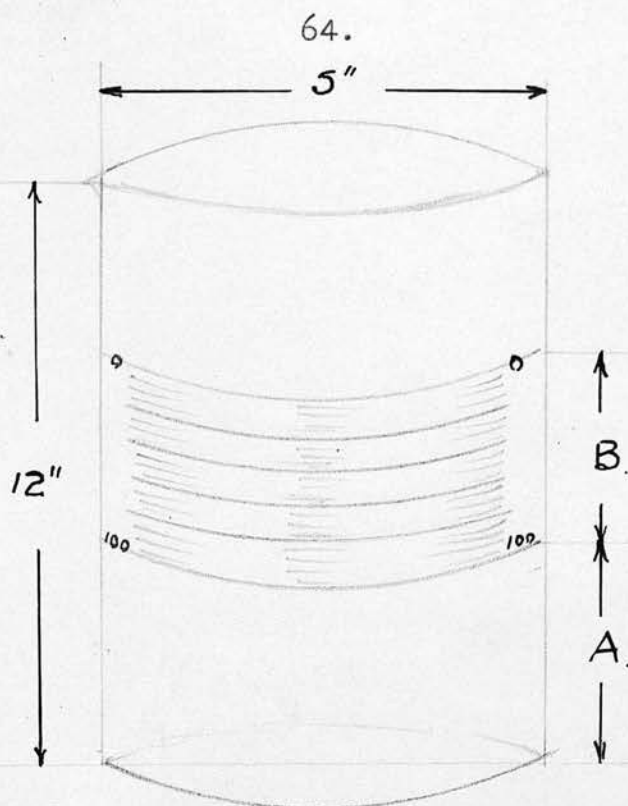


FIG. 14.

The cylinder is placed on a level surface and water poured in to fill part A. The material is then added until it fills part A, the vessel being gently tapped on its support so as to compact the material. The water surface will rise to, say, C, dividing part B into two portions: (a) that filled by water displaced by the material and (b) the upper part representing the voids. With the scale graduated as shown, the percentage of voids can be read direct. When using material which froths or clouds the water, a quick determination can be made by first packing the material to the level A and then adding 1,500 cc. water, this being the volume of that part of the vessel. The voids are again read direct. The

following results were obtained by using the above method, and are selected as typical of numerous tests:

TABLE 8

Size	Percentage of Voids		
	Colliery A	Colliery B	Colliery C
	White sandstone	mixed shale & sandst.	Shale
-3+2"	48	46	50
+1"	52	50	52
+ $\frac{1}{2}$ "	46	48	46
+ $\frac{1}{4}$ "	40	42	42
- $\frac{1}{4}$ "	28	34	32

For comparison the following table gives void percentages of various materials of different sizes, grouped in descending order:

TABLE 9

Material	Size	Percentage of Voids
Slag . . . . .	3"-2"	48
Granite chippings . . . . .	$\frac{3}{4}$ "	47
Furnace clinker . . . . .	1 $\frac{1}{2}$ "	46
Crushed stone . . . . .	$\frac{3}{4}$ "	45
Crushed sandstone . . . . .	$\frac{1}{2}$ "	44
Granite . . . . .	$\frac{1}{4}$ "- $\frac{1}{8}$ "	43
Sandstone . . . . .	2"	42
Quartzite . . . . .	1"	41
Limestone . . . . .	3"-1"	41
Grit . . . . .	1"- $\frac{1}{4}$ "	41
Sandstone. . . . .	1"- $\frac{1}{4}$ "	39
Limestone. . . . .	1 $\frac{1}{2}$ "- $\frac{1}{2}$ "	38
Broken Brick . . . . .	1 $\frac{1}{2}$ "	38
Round Pebbles . . . . .	3 $\frac{1}{2}$ "-1 $\frac{1}{2}$ "	37
River Sand . . . . .	- $\frac{1}{4}$ "	34
Gravel . . . . .	$\frac{3}{4}$ "- $\frac{1}{8}$ "	32
Sand (sea shore). . . . .	- $\frac{1}{8}$ "	31
Mixed Gravel and Sand . . . . .	- $\frac{3}{4}$ "	28

It will be seen that there is no direct connection between the size of the material and the percentage of voids.



This is owing to the varying quantity of different sized pieces or grains as well as their shape. For this reason two materials which come within the limits of the same riddles or sieves may yet contain different percentages of voids. Each material, therefore, should be judged and tested upon its merits.

#### Void Tests of Actual Stowing Material

In order to get information on the void percentage of material used in stowing, tests were made on materials from different sources. Size tests were first made and the figures given for void percentage are the average of a number of tests.

Material I: Washer Redd.

Size Analysis	3-2"	17.4%
	2-1"	31.8%
	1- $\frac{1}{2}$ "	26.3%
	$\frac{1}{2}$ - $\frac{3}{4}$ "	15.7%
	$\frac{3}{4}$ - $\frac{7}{8}$ "	6.3%
	- $\frac{7}{8}$ "	2.5%

Void % 40%

Material II: Crushed Shale (1)

Size Analysis	3-2"	11.7
	2-1	54.1
	1- $\frac{1}{2}$ "	20.0
	$\frac{1}{2}$ - $\frac{3}{4}$ "	8.3
	$\frac{3}{4}$ - $\frac{7}{8}$ "	3.5
	- $\frac{7}{8}$ "	2.4

Void % 45%

Material III: Mixed Table Pickings

Size Analysis	-3+2"	13.3
	2-1	42.8
	1- $\frac{1}{2}$ "	13.5
	$\frac{1}{2}$ - $\frac{3}{4}$ "	10.1
	$\frac{3}{4}$ - $\frac{7}{8}$ "	9.6
	- $\frac{7}{8}$ "	10.7

Void % 42%

## Material IV: Crushed Shale (2)

Size Analysis	3-2"	8.2%
	2-1	32.0
	1- $\frac{1}{2}$	28.6
	$\frac{1}{2}$ - $\frac{3}{4}$	9.9
	$\frac{3}{4}$ - $\frac{1}{8}$	12.5
	- $\frac{1}{8}$	8.8

Void % 48%

From the tests, the void percentage is shown to be between 40-48%. It was visually apparent from the material that the washer redd packed more compactly, although the smaller fractions were of a low percentage. The shale samples were long shaped and packed loosely, although the  $\frac{1}{2}$ " fraction was higher.

#### Minimum Voids

Together with particle shape, the amount of small material in the general body of the material has an important effect on the void percentage. The packing of the inter particle spaces by small material compacts and solidifies any resultant pack. Tests were made by running measured quantities of fine sand into the actual stowing material samples in order to determine the effect on void percentage - in other words, to see whether any information could be obtained on the percentage of small material to be added or to be contained in a given material to get minimum voids.

A test on sand alone gave the figure of 33% voids. It must be remembered that damp sand or small sized material bulks or occupies more space than it

does when completely dry. Moist sand containing 2 to 5% of moisture by weight, will increase the volume by 15-30% or even 40% as compared with the same material when bone dry. Increase of the moisture content to above 5 or 6% gradually reduces the bulking, until sand that is completely saturated has almost the same volume as when bone dry.

TABLE 10

Material	original Voids %	Bulk vol.	Sand added cc.	Resultant Voids %
I	40	1500	550	20
II	45	"	460	25
III	42	"	390	24
IV	48	"	500	22

A similar test was carried out on the sieved fractions of a shale packing material. The material was placed in a void test jar and levelled to the 1500 cc. mark. Sand was run in until all inter-spaces were filled, the jar being shaken gently to ensure complete filling. The quantity of sand was measured in cubic centimetres. The deduction of this figure from 1500 will give the approx. bulk of stowing material in the jar, and hence the percentage of the ingrediedients.

TABLE 11

Size	Original Voids %	Bulk Vol.	Sand added cc.	Resultant Voids %
-3+2	50	1500	710	18
2+1	52	"	780	16
1+ $\frac{1}{2}$	44	"	760	16
$\frac{1}{2}$ + $\frac{1}{4}$	42	"	660	17
- $\frac{1}{4}$	32	"		

It will be seen from Table 10 that the addition of sand considerably reduces the void percentage in stowing material mixtures, and that the addition required is approx. 30%. Taking the added fines and the fines below  $\frac{1}{4}$ " in the material together, it shows that for maximum density the stowing material should contain in the region of 35-45% of small sized material.

Table 11 gives the results of interfilling with sized material and shows that more small sized fractions can be added or are absorbed giving a resultant mixture of less void percentage. This would appear to show that stowing material of close sizing is not as suitable for underground packs as suitably-mixed material. The degradation of material whilst in transit may, therefore, have a beneficial effect upon the resistance characteristic of the pack, by causing a reduction in the voids. These tests show that theoretically more efficient roof support should result in the use of small sized material for pack building.



One very important feature disclosed in these tests is the high percentage of voids to be found in materials used for underground packing. Even with efficient hand packing it is probable that the void percentage is in the region of 40-50%, which means that as far as immediate roof support is concerned packs are of little value. With mechanised stowing an important development must be the consideration of means to pack the material tight either by increasing the speed of propulsion into the goaf, or by some ramming arrangement.

Taking a wider view of this subject, it is apparent that the present methods of hand packing, in the majority of cases, cannot possibly be efficient, and can only be regarded as a means of dirt disposal until such time as the movement of the roof and floor gradually compress the material until, by the elimination of voids some resistance is offered at a very late stage in the operations. In view of the doubtful value of hand packing as a means of roof control and the tremendous cost involved, it is of extreme importance that this phase of underground mining should be critically examined, by the management of collieries where hand packing is practised.

(ii) Compression Tests

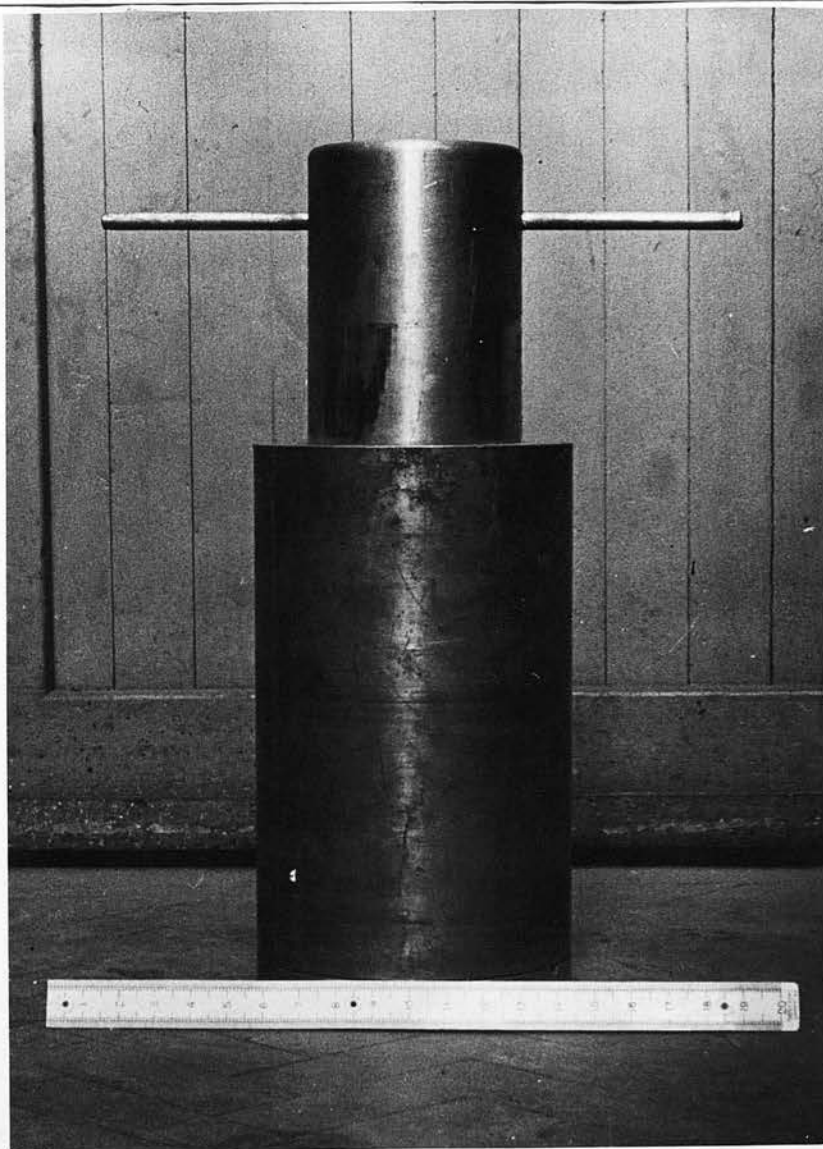
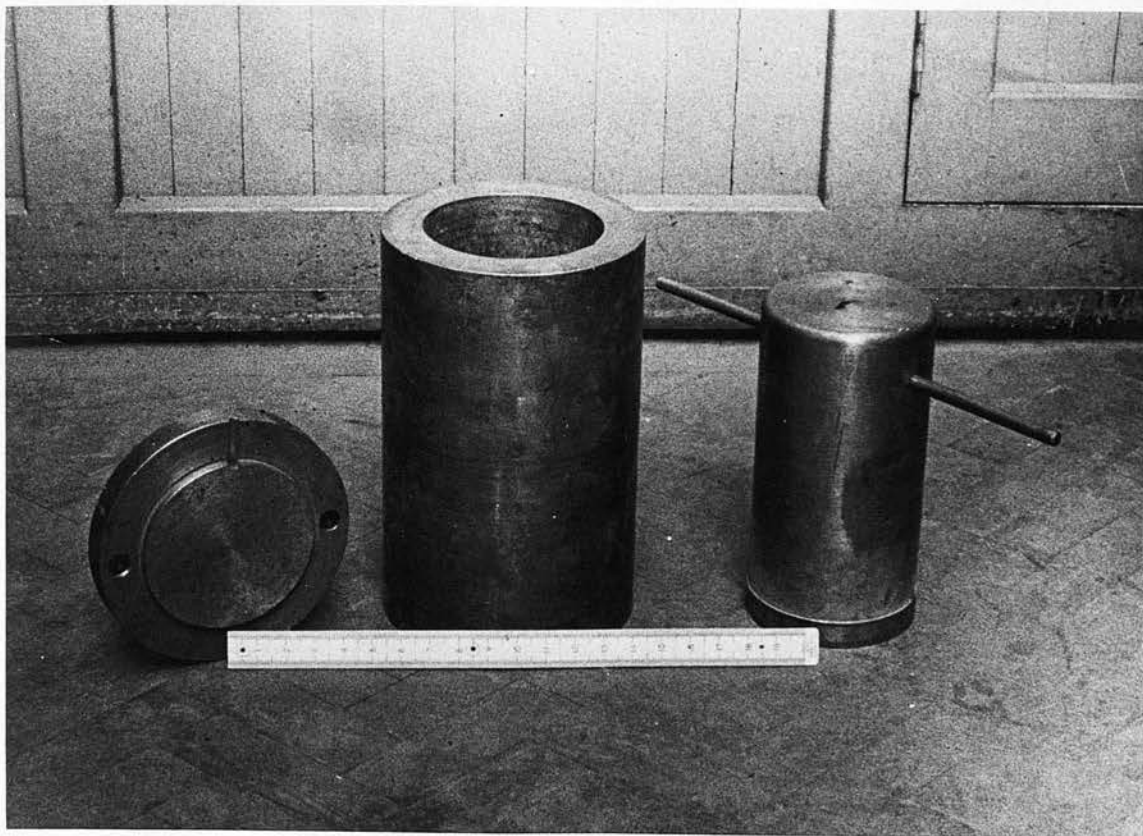
The degree of support afforded by packing depends largely on the compressibility of the material used and plays an important part in roof control and extent of surface subsidence. The need of a standard test to evaluate the suitability of materials available for the construction of packs is essential in view of the increased use and probable large-scale development of mechanised packing or stowing. Such tests, especially of compressibility, would give some indication of the degree of resistance to roof pressure and it would be possible to ascertain with some degree of accuracy the probable amount of subsidence to be expected under given circumstances, as the behaviour of the roof, coal and floor material under forces operating in mines will depend on their relative strengths under these forces.

Laboratory tests on actual packs showed that packs failed at relatively small amounts of compression and load, whereas underground packs seem to stand considerable pressure and compression.\* It is supposed that the resistance of packs underground is increased by lateral support - by penetration of the floor and pressure of supports. The tests showed that the compression increased with the load up to the point of collapse, and if collapse is

\* "Tests on Packs", L.J. Barraclough, S.W. Inst. Engrs., 1934.

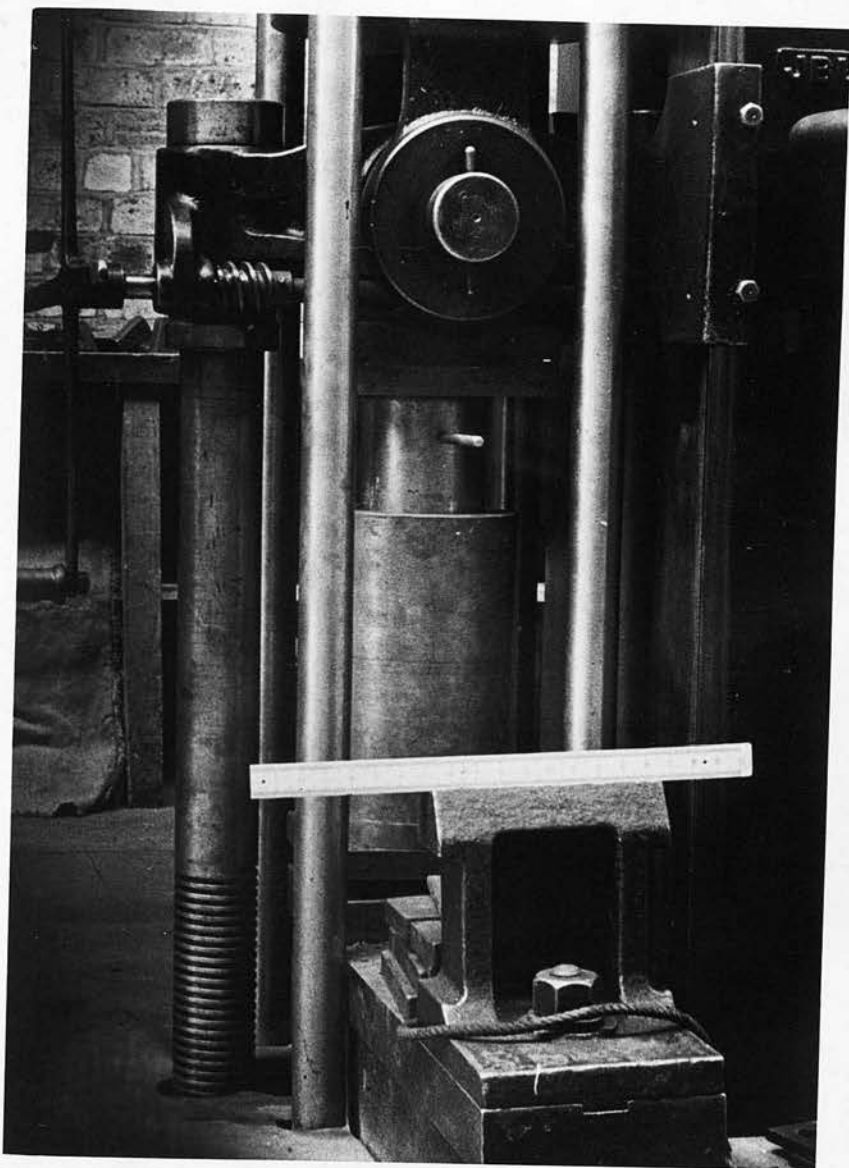
delayed so that the pack undergoes more compression, then it was deduced that the resistance would be proportionately increased. Lateral support is a main factor and if non existant at the beginning of pressure, can be built up from within the pack by the self-locking action of materials under compression. Whilst the results obtained by testing packing materials for compression in a barrel and plunger apparatus may be considered theoretical, by reason of the complete lateral support given by the sides of the barrel, they can give an indication of the ultimate behaviour of the material. It is only necessary to determine the initial compression or flow of a pack under roof pressure to deduce the final maximum compression. The conditions obtaining in a pipe sample area in the middle of an initially compressed pack will be very near the laboratory conditions.

A considerable number of tests were carried out on different materials and differing sizes of the same material. It was decided to use two similar sets of equipment differing only in barrel length. The tests were done firstly in a steel cylinder 6" diam., 14" long, having a 6" diam. ram and a second set in a similar cylinder 4'8" in length. A photograph of the smaller cylinder and plunger is shown in Fig. 15 while Fig. 16 gives a view of the apparatus in the testing machine. (Buckton 100 ton testing machine.)



- FIG. 15. -



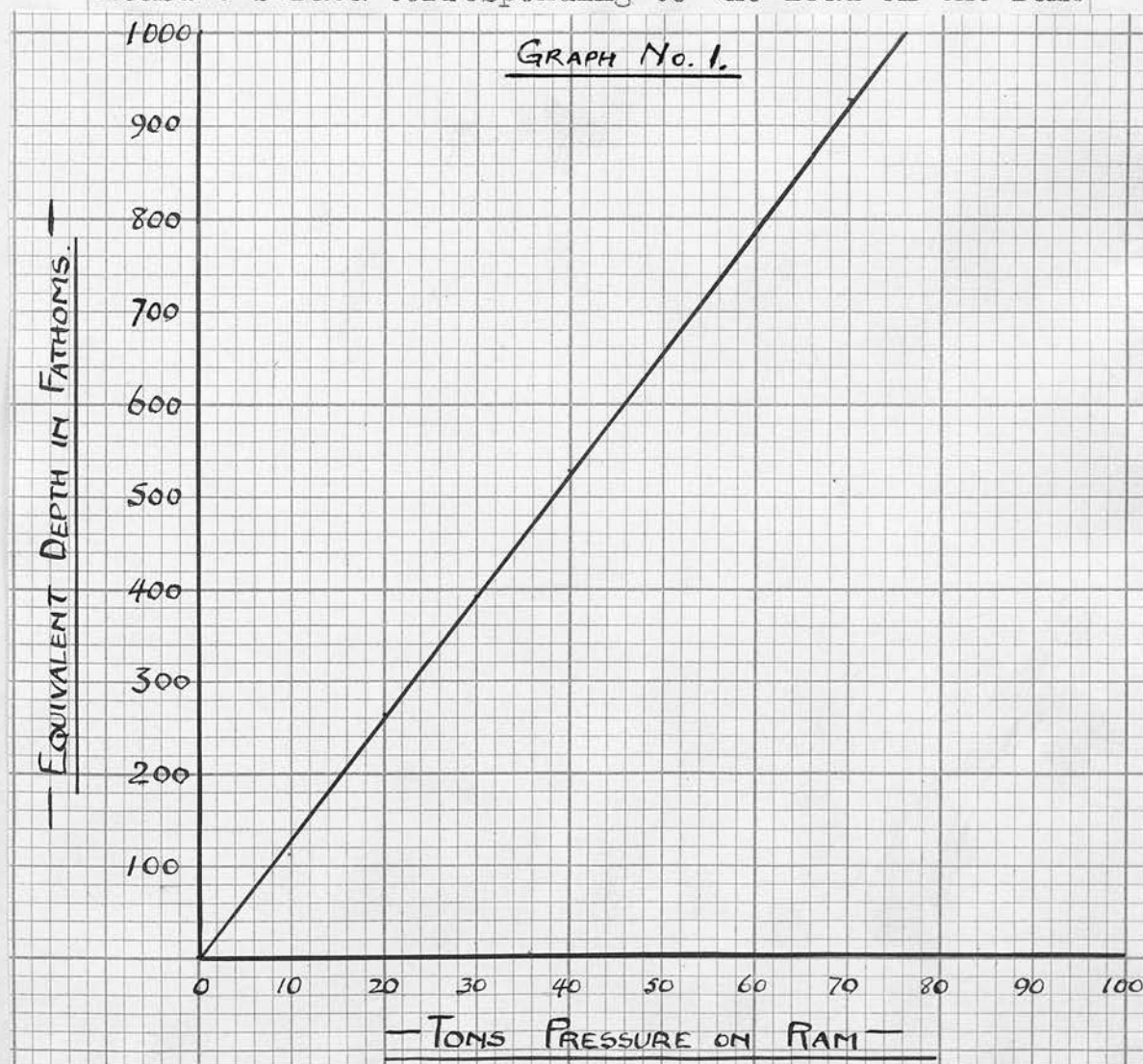


- FIG. 16 -

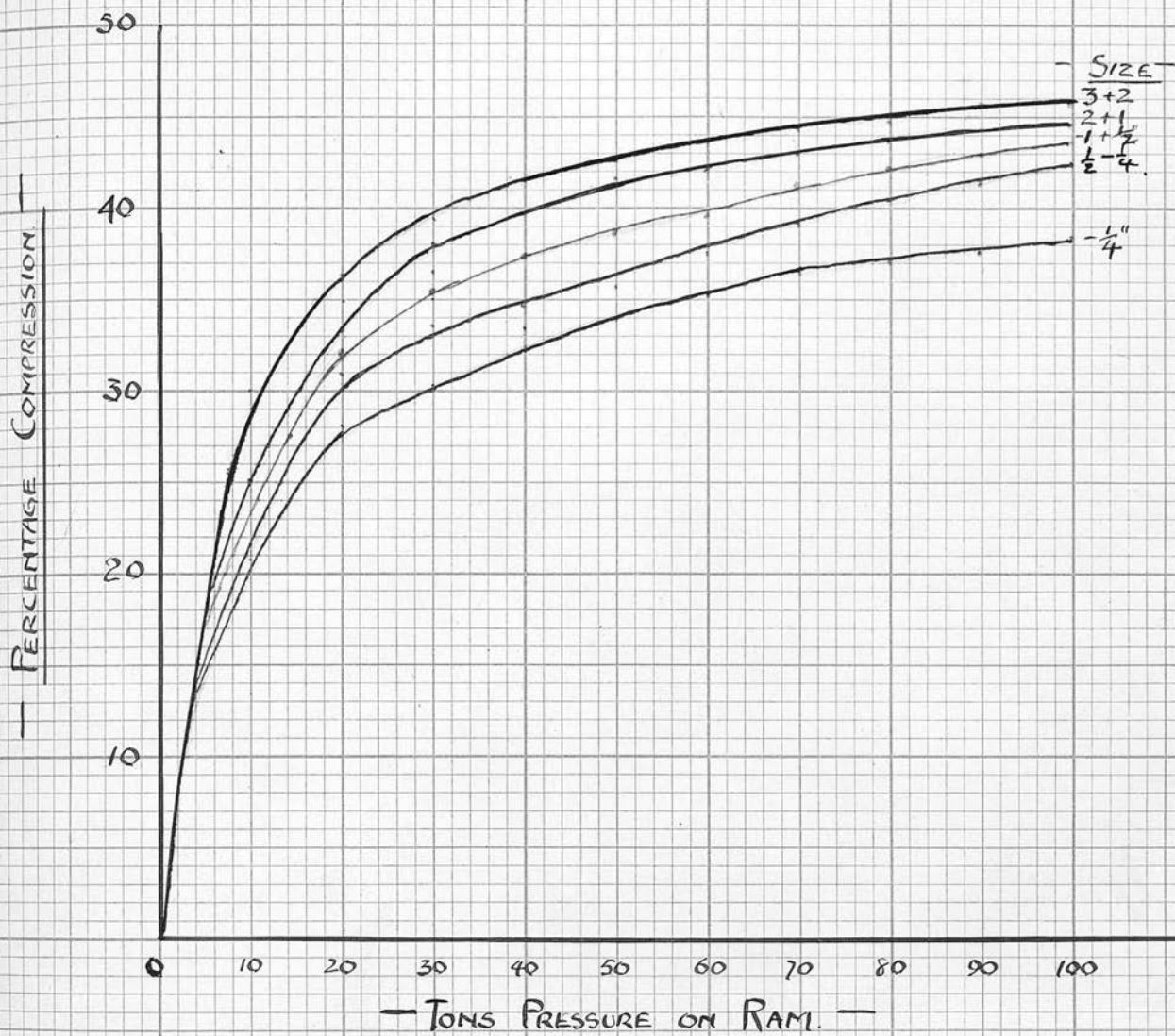
The material to be tested was placed in the barrel and gradually put under load in increments up to 100 tons on the ram, and the compression noted. The equivalent tons and pounds per square inch were as follows:-

<u>Pressure in Tons</u>	<u>Pressure in lbs./sq.ins.</u>
2 . . .	158.4
10 . . .	792
20 . . .	1584
30 . . .	2376
40 . . .	3168
50 . . .	3960
60 . . .	4752
70 . . .	5544
80 . . .	6336
90 . . .	7128
100 . . .	7920

Graph No. 1 gives the equivalent depth of coal measure strata corresponding to the load on the ram.



GRAPH No 2



— MATERIAL - SHALE —

Test Results -- in the first series of tests shale and sandstone were sized into fractions from  $-3"$  to  $-\frac{1}{4}"$  and each fraction tested separately. Table I2 gives the results obtained for a typical coal measure shale and Table I3 the results for a white sandstone. The compression is given as a percentage of the original length of material in the barrel. The following tests were conducted in the small ( 12") barrel.

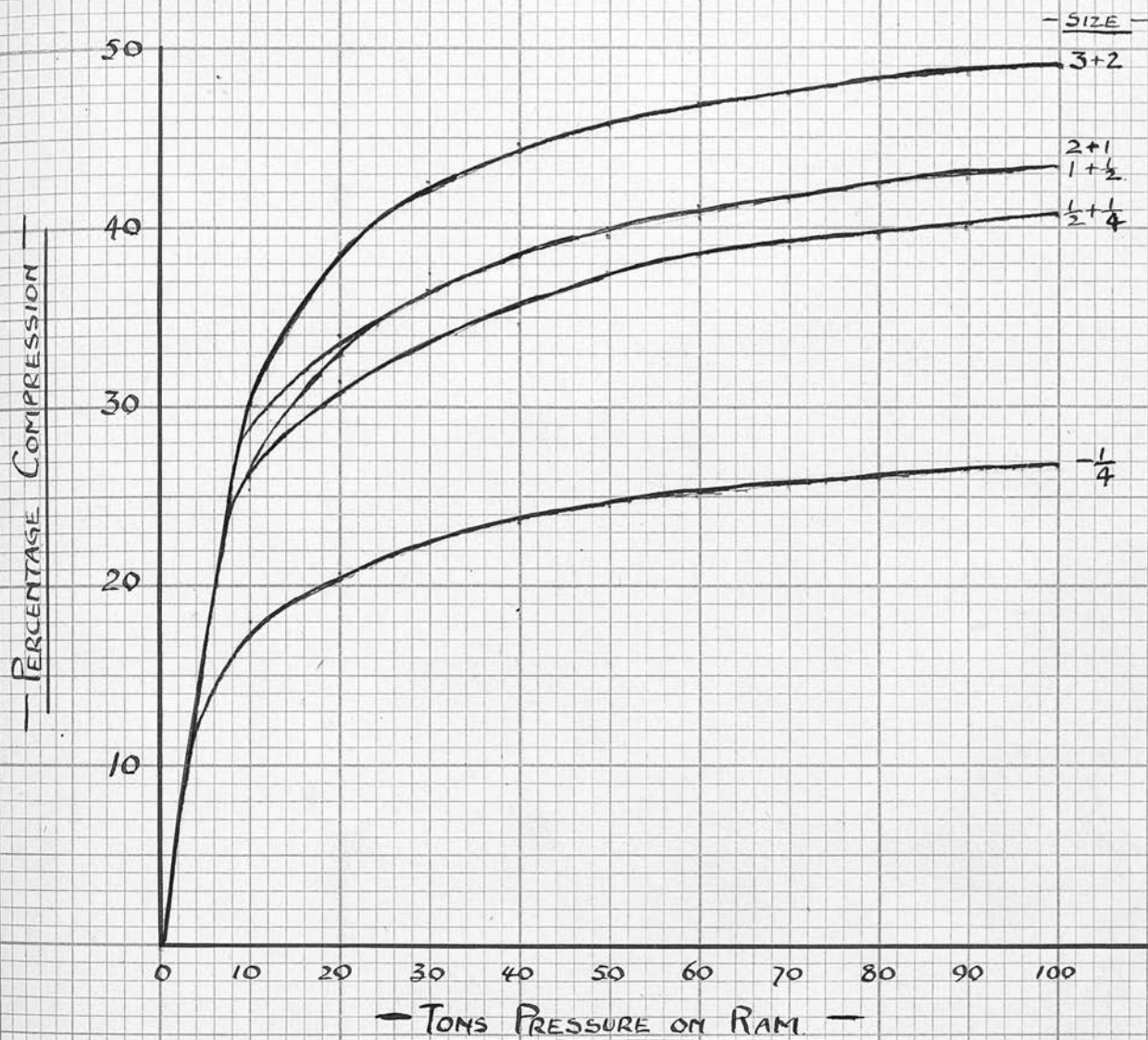
TABLE I2 (Shale)  
(Graph No.2)

Press. in Tons	Press. lbs.per sq.ins.	Percentage Compression				
		Size				
		$-3+2$	$2+1$	$1+\frac{1}{2}$	$\frac{1}{2}+\frac{1}{4}$	$-\frac{1}{4}$
2	158.4	12.5	14.2	15.2	18.4	16.0
10	792	29.0	25.0	26.1	20.6	21.6
20	1584	36.4	33.3	32.2	31.0	28.1
30	2376	39.5	38.0	35.5	33.3	30.2
40	3168	41.7	39.6	37.2	34.8	33.4
50	3960	42.9	41.7	38.1	35.9	34.0
60	4752	43.7	42.0	39.5	37.6	35.1
70	5544	44.2	43.0	41.3	39.1	36.4
80	6336	44.7	43.8	42.1	40.6	37.0
90	7128	45.8	44.2	43.2	41.3	37.5
100	7920	45.8	44.7	44.0	42.4	38.2

From the table it will be seen that in all size fractions the major part of the compression occurs before one third of the load has been applied. The compression rate is greatest with the initial loads and the increase in compression is much reduced after the material has been compressed to approx. 40%. It



GRAPH No 3



MATERIAL - SANDSTONE

appears that there is little resistance to compression until the material has been reduced to approx. 60% of its original volume. It will also be noticed that the initial compression increases with size reduction, but that the total compression decreased with size reduction.

TABLE 13 (Sandstone)

Press. in Tons.	Press. lbs.per sq.ins.	Percentage Compression				
		-3+2	2+1	Size	$\frac{1}{2}+\frac{1}{4}$	$-\frac{1}{2}$
				$\frac{1}{2}$		
2	158.4	15.3	13.4	12.6	8.3	11.2
10	792	30	33	26	25.5	18.1
20	1584	38.7	34	33	31.25	20.8
30	2376	42.6	36.4	37.5	33.8	22.4
40	3168	44.75	38.6	38.5	34.8	23.4
50	3960	45.8	39.5	40.2	37.5	24.4
60	4752	47.0	40.6	41.1	38.5	25.0
70	5544	47.25	41.7	41.7	39.0	25.5
80	6336	48.1	42.1	42.3	39.5	26.0
90	7128	48.4	42.6	42.8	40.0	26.5
100	7920	49.0	43.2	43.3	40.6	26.5

This sandstone shows slightly less compressibility than the shale, especially in the smaller size fractions. In general the percentage of compression decreases with decrease in size. Sandstone breaks down to sand the filling of the inter spaces increasing the resistance to compression. Here again it is noticeable that the compression is greatest under the first light loads and then diminishes. The final compression is approximately the same as for

shale with the exception of the  $-\frac{1}{4}$ " fraction, which can be regarded as sand. For interest, ordinary sea shore sand of fine quality was tested under similar conditions and gave the following results:

TABLE 14 (Sand)

<u>Press.Tons.</u>	<u>Press.lbs/Sq.ins.</u>	<u>Compression %</u>
2	..... 158	..... 1.3
10	..... 792	..... 3.9
20	..... 1584	..... 5.9
30	..... 2376	..... 7.9
40	..... 3168	..... 9.25
50	..... 3960	..... 10.5
60	..... 4752	..... 11.8
70	..... 5544	..... 13.1
80	..... 6336	..... 14.4
90	..... 7128	..... 15.1
100	..... 7920	..... 15.7

Sand is known to have little compression, but this compression is spread evenly over the increase in load. There is no rapid increase even at the first compressing loads. This may be due to inter particle binding, a characteristic not apparently found in other material. This factor is clearly shown when considering percentage voids against ultimate compression percentage.

Table **15** shows the results of tests conducted on material used for pneumatic stowing at a Scottish colliery. The sample was crushed shale mixture and the size analysis is as below:-

-3	2	...	13.3%
-2	1	...	42.8%
-1	$\frac{1}{2}$	...	13.5%
$-\frac{1}{2}$	$\frac{1}{4}$	...	10.1%
$\frac{1}{4}$	$-\frac{1}{8}$	...	9.6%
	$-\frac{1}{8}$	...	10.7%

TABLE 15 (Crushed Shale)

<u>Pressure in</u> <u>Tons.</u>		<u>Pressure in</u> <u>lbs./sq.ins.</u>		<u>Percentage</u> <u>Compression</u>
2	.....	158.4	.....	11.0
10	.....	792	.....	23.0
20	.....	1584	.....	28.0
30	.....	2376	.....	35.4
40	.....	3168	.....	35.8
50	.....	3960	.....	36.4
60	.....	4752	.....	36.7
70	.....	5544	.....	37.5
80	.....	6336	.....	38.0
90	.....	7128	.....	38.5
100	.....	7920	.....	39.0

The material shows a steady compression with load and at the load corresponding approximately to the strata pressure indicates a compression in the region of 30%. The mixture was well graded for size, the initial compression only showing 11%. The ultimate compression percentage of 39% must be regarded as satisfactory, although this could be brought down considerably by the addition of material of below,  $\frac{1}{4}$ " size.



The use of the 4' 8" testing cylinder was confined to a 50 tons Greenwood & Batley compression machine, as the overall length of the barrel and plunger was approximately 6' 0". These tests were conducted on the same material as tested in the 12" cylinder, but only up to half the pressure range.

Table 16 gives the results of a test on material used for stowing at a Scottish Colliery, the size analysis being approximately the same as used in the previous test (page 76 , Table 15).

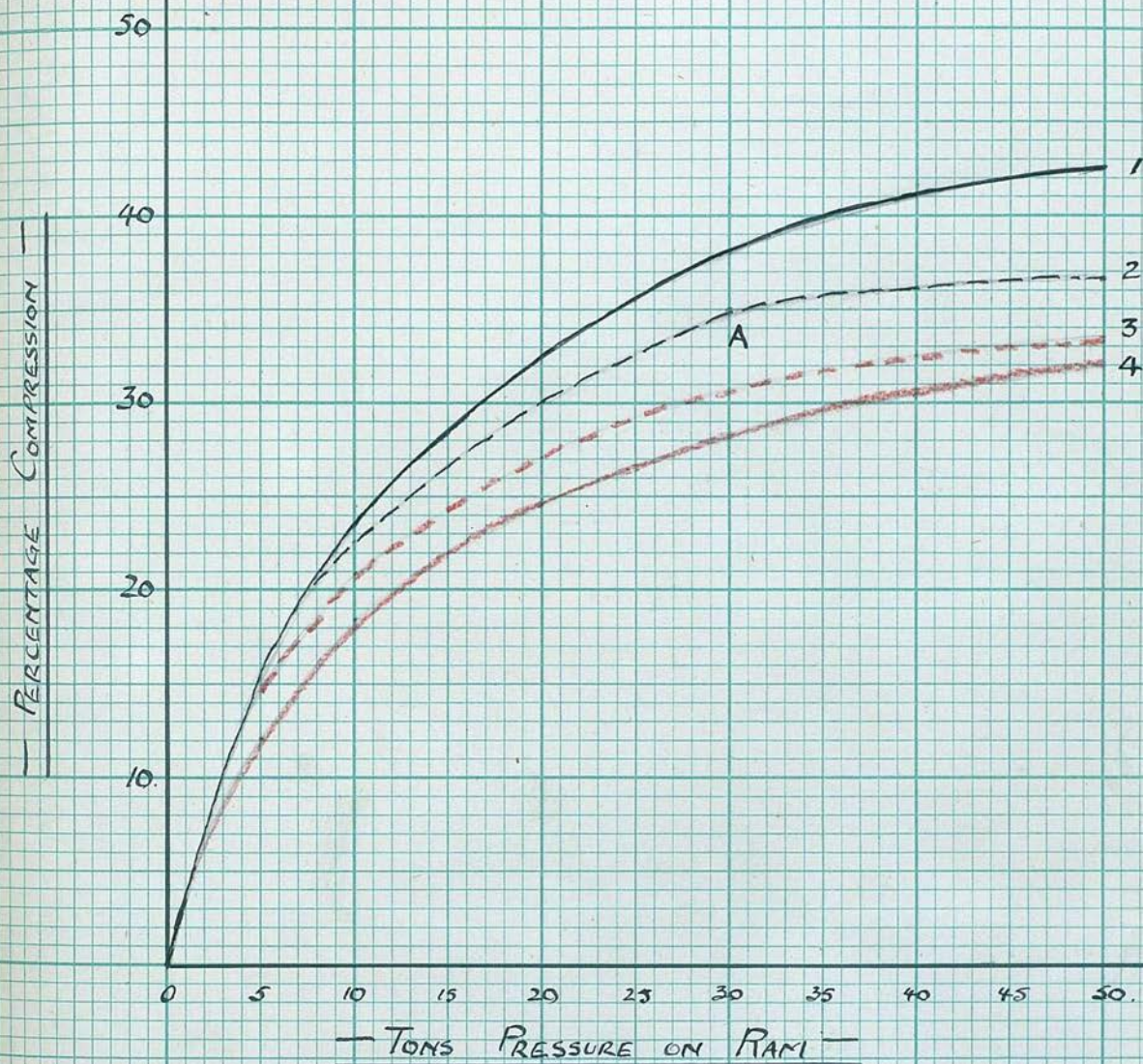
TABLE 16

Pressure in Tons	Pressure in lbs.per sq.ins.	Percentage Compression	
		3'10"depth	1'10"depth
10	792	23.9	23.1
20	1584	31.7	28.1
30	2376	38.3	35.4
40	3168	40.5	35.8
50	3960	42.7	36.4

The above results are plotted on Graph No.4 and show that up to 10 tons the percentage compression is approximately equal. It would appear that the height of pack does not affect the percentage compression during the initial pressures and as shown on the graph the divergence increases as the load is applied. As the material is compressed, the stage of effective resistance to compression is reached at a lower load with a thin pack than with a pack of greater height. Referring to the graph, the material at 12" depth is offering effective



# GRAPH No 4



CURVE No 1 - CRUSHED SHALE - 3'-10" DEPTH.

CURVE No 2 - CRUSHED SHALE - 1'-0" DEPTH.

CURVE No 3 - MIXED TABLE PICKINGS

CURVE No 4 - SANDSTONE.

resistance at point A (30 tons), whereas the material at 3'10" depth is still compressing uniformly.

Sandstone packing material and mixed table pickings and washery refuse were tested in the 4'8" cylinder for comparison.

TABLE 17

Pressure in Tons.	Pressure in lbs.p.sq.ins.	Percentage Compression	
		Sandstone	Mixed table pckngs
5	396	12.20	15.76
10	792	19.40	21.2
15	1188	22.80	25.0
20	1584	25.00	27.1
25	1980	27.2	28.8
30	2376	28.3	29.9
35	2772	29.0	31.0
40	3168	29.4	32.1
45	3564	31.1	33.1
50	3960	32.2	33.7

With the depth of material tested there is little difference in the resistance to compression between the two materials. The maximum compression up to the full test load was 32.2% for sandstone as against 33.7% for the mixed refuse. The compression graph shows the standard type of curve, the compression difference being constant after the initial load of 5 tons. (GRAPH No. 4.)



Compression and Void Percentage - In order to try and correlate the ultimate compression and the percentage of voids in the original material, the main factors were tabulated as shown in TABLE VII.

TABLE 18

Material	Size	Initial Load	Initial %age Compress	Final Load	Final Compress %age	Original Void Percentg
SHALE	-3+2	158.4 lbs/sq. ins.	12.5	7929 lbs per sq. in.	45.8	50
	-2+1	"	14.2	"	44.7	52
	-1+ $\frac{1}{2}$	"	15.2	"	45.8	46
	- $\frac{1}{2}$ + $\frac{1}{4}$	"	18.4	"	42.4	42
	- $\frac{1}{4}$	"	16.0	"	38.2	32
SAND-STONE	-3+2	"	15.3	"	49.0	48
	-2+1	"	13.4	"	43.2	52
	-1+ $\frac{1}{2}$	"	12.6	"	43.3	46
	- $\frac{1}{2}$ + $\frac{1}{4}$	"	8.3	"	40.6	40
	- $\frac{1}{4}$	"	11.2	"	26.5	28
STOWING MATERIAL	Mixed	"	11.0	"	39.0	42
SAND		"	1.3	"	15.7	33

The results show that within the limits of experimental error the final compression of a sized material approaches closely to the figure of the original void percentage of the material. In the above examples, the  $-\frac{1}{4}$  fraction in shale shows the only result where the final compression is in excess of the original void percentage. This may be explained



by the observed condition of the material at the end of the compression test. Under such a load, the fine shale became almost solid, much more so than any of the other samples. The larger fraction of the sandstone samples showed the highest final compression, due probably to a greater degree of point contact than the corresponding size in shale. The smaller sized sandstone fractions, however, exhibit a greater resistance than their shale counterpart, this applying to both initial and final compression results. The mixed stowing material with an original void content of 42% had a final compression of 39%, which is less than the compression of a sized material of the same void content.

It is obvious that at some stage in the compression factors other than the void content play an important part in the resistance to pressure. The crushing strength of the material, parallel and at right angles to the stratification will be the criterion at high pressures, whilst the liability to disintegrate and thus compact the interstices will help to resist compression at lower pressures.

8. MATERIAL HANDLING

(a) PREPARATION AND SUPPLY OF STOWING MATERIAL.

(b) TRANSPORT AND FEED ARRANGEMENTS

- (i) Pneumatic Stowing.
- (ii) Mechanical Stowing.

MATERIAL HANDLING(a) PREPARATION AND SUPPLY OF STOWING MATERIAL

A maximum size of 3" is generally accepted as a suitable size material for mechanised stowing purposes, and the following types are in use: washery shale, screen pickings, pit debris, boiler ashes and pit-tip material.

The position of the preparation plant may be on the surface or underground, depending upon the main source of supply. Where the bulk of the material is to be supplied from a washery plant or pit tip, the plant is located at a convenient point on the surface, whereas an underground plant is preferred for pit debris, to avoid the need for winding the material. The extent of layout of the preparation plant may require considerable excavation, and unless this is made in hard rock, may prove to be an expensive item, especially where roof pressures are excessive. In general, provided the shaft winding capacity is adequate, the stowing material preparation plant is best located on the surface.

The layout and method of preparation is standard wherever the plant is placed. Briefly, the method commonly adopted is as follows: material is fed by a tippler on to an inclined picking table where oversize lumps, tramp iron, etc., can be removed. This conveyor delivers into either a shaking or rotary screen, the undersize being conveyed to a storage bunker and the oversize into a crusher. Sometimes

the crushed material is re-screened. The most important piece of equipment in the preparation of material is the crushing machine, as the resultant size and shape of the finished product is of paramount importance. It will depend upon the structure of the stowing material as to which type of crusher will give the best results. The types of crushers in use can be divided into three groups:

1. Jaw Crushers (Plain bearing and Roller bearing)
2. Swing Hammer Crushers.
3. Gyratory cone Crushers.

The Jaw crusher is a common type of crusher but for stowing material preparation has the disadvantage of producing long flat pieces. It can, however, handle fairly large material and if recirculation of the product is practiced this type can be used successfully. Another point is that for high capacity the machine is large and costly. The design and development of the high speed roller bearing jaw crusher has offset many of the above disadvantages.

The Swing Hammer Crusher (Fig.17) appears to be the most suitable for material preparation, as it is the most compact for its capacity and gives a cubical product. The crushing action of this machine depends upon the impact of a series of hammers on the material to be broken. The rotor speed is varied according to the hardness of the material, and may vary from 450-700 R.P.M. Owing to its small size and weight, and the vibration-free nature of the



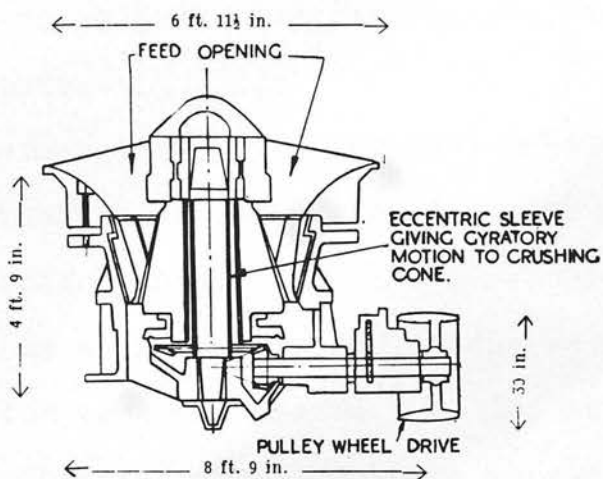


FIG. 18.—PEGSON TELSMITH 13B GYRATORY CONE CRUSHER.

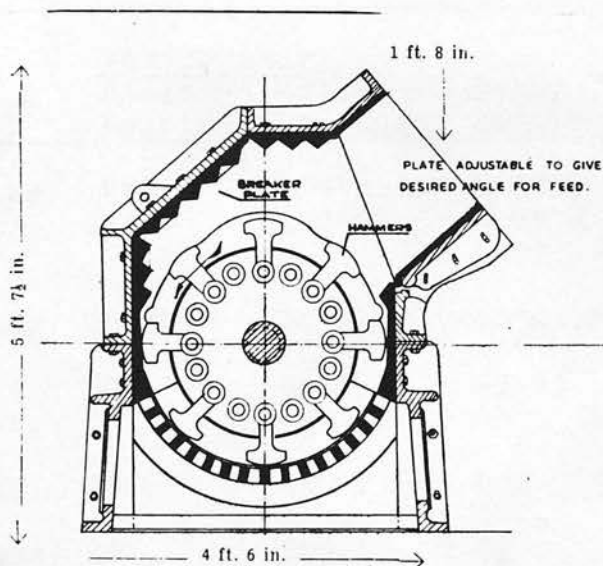


FIG. 17.—MANSFIELD HAMMER CRUSHER (NO. 3 HEAVY TYPE).

crushing, it can be mounted on high staging with safety. The main disadvantages are the high power cost and limited feed size. These machines are, however, being given serious consideration in new plant layouts.

In Germany, the Gyratory cone type of crusher is extensively used because of its high capacity and compact design, (Fig.18). The gyratory motion is generated by a driven eccentric sleeve between the cone and the vertical shaft. It is, however, larger and heavier than the swing hammer type, but has less power consumption for a rated capacity.

A summary of the main features of the three types is given in Table 19.

TABLE 19.

	<u>Jaw Crusher</u>		<u>Swing Hammer</u>	<u>Gyratory Cone.</u>
	<u>Plain Bearing</u>	<u>Roller Bearing</u>		
Overall size	9x8'x 12'6"	7'6"x11' x 7'6"	5'8" x 8'6" x 4'6"	7'7' x 8'6"
Feed opening	36"x24"	30"x18"	40"x19"	59"x13"
Capacity in tons per hr.	50	50	50-70	80
Horse Power	85	60	100	50
Weight (Tons)	28	19½	7½	13
Cost (1949) £	4,500	2,838	1,700	2,500
Type of Product	Flat	Flat	Cubic.	Mixed.

Bunker Arrangements - Bunker capacity for prepared material is a fundamental necessity, as with

out such arrangements, much loss of stowing time can be caused by material shortage. Where the material is to be sent down a shaft in trams, the best surface arrangement is to have a bunker of a capacity at least equal to a shift of stowing. The bunker should be so placed that the trams can be filled direct whilst in circuit on the empty side of the shafts. A magnetic separation pulley should be incorporated in some part of the circuit, preferably before the crushing unit. Magnetic separation is particularly important in pneumatic stowing installations due to the serious damage that can be caused by tramp iron to the paddle blades of the stowing machine.

To illustrate successful layouts of crushing and preparation plants, a brief description is given of one underground and one surface installation.

Michael Colliery Underground Crushing Plant. (Fig.19)

In this scheme, tubs of dirt from other seams being worked from the same shafts are segregated at a convenient point near the pit bottom and taken by haulage to the crusher plant. The tubs are emptied by means of a revolving tippler on to a 3 ft. wide plate picking table, atomising sprays being fitted to the tippler. Tramp iron, coal, and other unsuitable material is picked off at this point. This plate conveyor delivers the stone on to a shaker screen which passes minus  $2\frac{3}{4}$  ins. material direct into a tub, whilst the oversize goes to a 36"x20" Baxter Breaker which has a rated capacity of 30/40

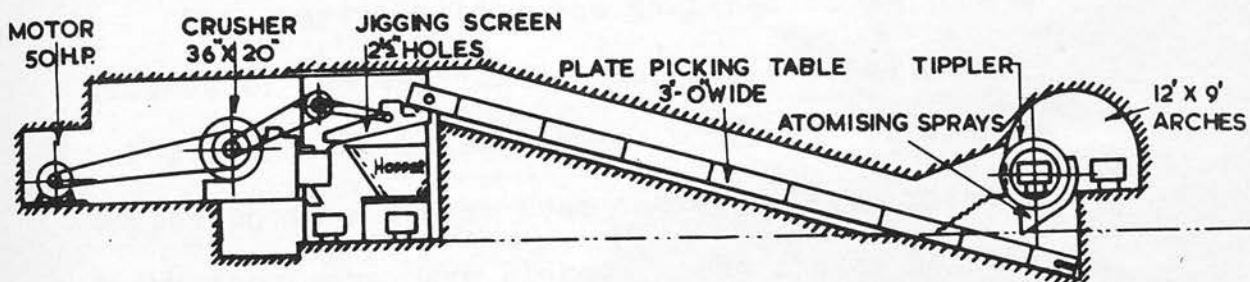


FIG.19 - Michael (Underground) Crushing Plant.

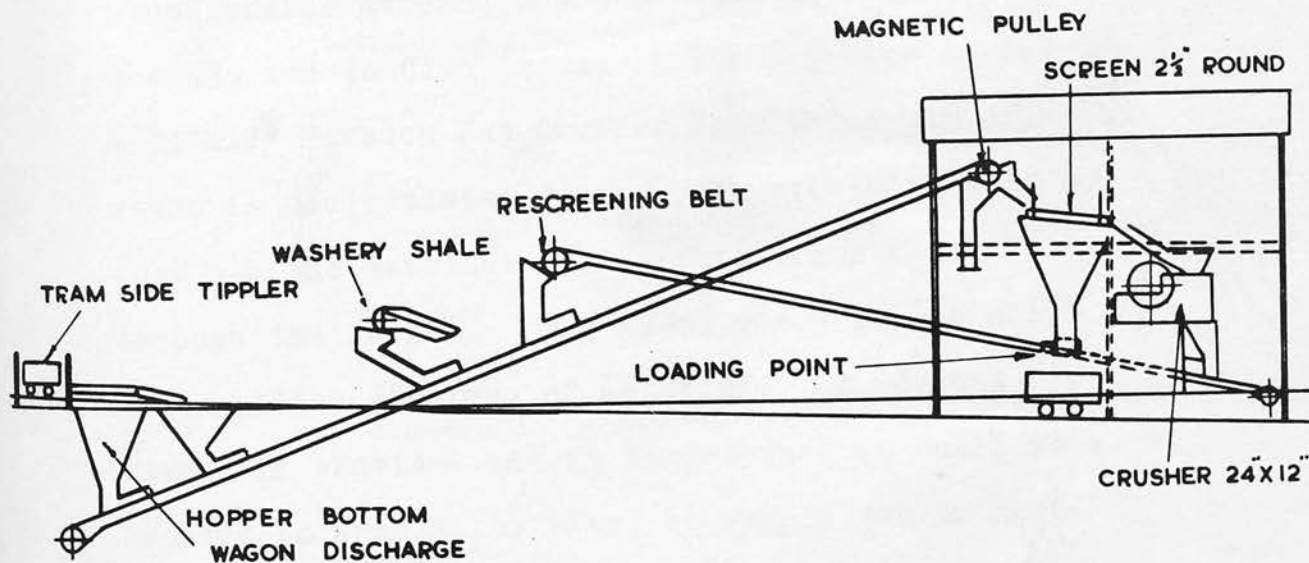


FIG.20 - Abercynon (Surface) Preparation Plant.



tons per hour down to  $2\frac{1}{4}$ . The breaker is driven by a 50 H.P. motor through a belt drive. To produce approx. 270 tons of prepared material, the above layout requires 16 men shifts.

Abercynon Surface Crushing Plant (Fig.20)

This surface plant was designed to produce a mixture of 50% washing debris, 30% crushed pit rubbish and 20% boiler ashes. The pit rubbish is tipped on to a shaker feed conveyor, from which coal, tramp iron, etc. are picked. The shaker delivers on to the main elevating belt which also receives the washery shale, via a belt direct from the washery and boiler ashes. The delivery end of the conveyor is fitted with a magnetic roller and feeds on to a  $2\frac{1}{2}$ " round shaker screen, the undersize of which falls to a chute for loading trams. The oversize is fed to a 24"x12" Marsden Jaw Crusher from which the crushed stone is recirculated to the main elevating belt so that all the material filled into trams passes through the screen. The plant works on one shift and supplies 250 tons of material. A 90 tons bunker is provided and it is proposed to build more bunkers to enable the plant to supply two or more stowing machines on a two-shift system.

(b) TRANSPORT AND FEED ARRANGEMENTS

The problem of efficient transport and feed arrangements are of vital importance both for pneumatic and mechanical stowing, as they determine whether or not power stowing can be successfully applied to any given set of conditions. In horizon systems of mining the solution is easier and presents few problems, as the stowing material can be taken in on an upper horizon that is free from the congestion of coal transport. The material can then be lowered by a staple pit or chute direct to the stowing machine or on to a feed conveyor in the supply gate. In conventional British Mining practice, where rope haulage is used, the transport of stowing material inbye amongst the empty tub supply is often the most serious obstacle in the introduction of power stowing. It cannot be too strongly emphasised that the success of power stowing depends more on haulage arrangements than on any other factor.

(i) Pneumatic Stowing

Generally speaking, the most convenient place for the installation of the pneumatic stower and tippler is in the tail or supply gate. The feed arrangements at the dumping point must be capable of feeding the stower at the rate of up to 3 tons per minute as this is the capacity of the latest types of stowing machines. Adequate siding capacity is advisable behind the tippler. The tippler may feed directly into the stowing machine or a short elevating conveyor can be used between

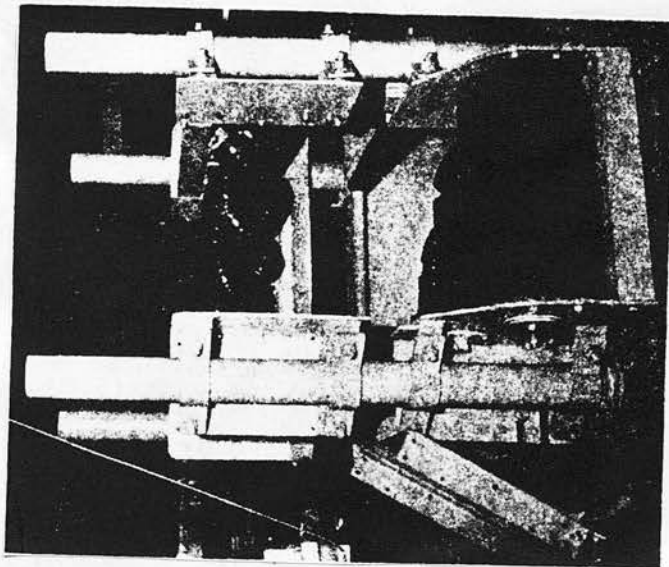
the tippler and the machine. An extending belt conveyor is also common practice. Where a short feeder is used, it has to be moved up with the machine, whilst with a belt, the tippler and sidings can be of a more permanent nature and the belt delivery point extended as the stowing machine moves forward. The present type of tippler in use is not entirely satisfactory and there is need for a compact design that will lift the tub higher from the floor and tip at a greater angle to obviate scraping out, and if sticky material is to be used, to give a bump at the end of the tipping cycle. The position of the stower and feed installation depends to a large extent on the type of stowing material used. If this is sticky and likely to cause blockages of the pipe, the distance from the face to the machine should be kept to a minimum and in practice is between 50 and 100 yards. With dry granular material this distance may be extended to 300-500 yards and thus a more efficient permanent station can be maintained well outbye. Scraper chain conveyors, shaker conveyors, and belts or combinations of these are used to supply the stowing machine, but where the material can be directly supplied to the machine from an overhead bunker, it is undoubtedly the best arrangement from a trouble-free operating point of view.

(ii) Mechanical Stowing

As compared with pneumatic stowing, where the stowing machine is situated in the supply gate, the feed to a mechanical stower is a much more complicated and difficult operation. It is this disadvantage of having to transport and feed the machine on the coal face that has retarded the introduction of this form of stowing. The material has to be tipped on to a gate belt, then transferred to the face belt and so down the face to the machine. With a daily turnover of the face belt and extension of the gate belt, the time available for stowing is limited, as in the present arrangements stowing and coaling cannot go on at the same time. The machine has to traverse the face and failing the solution of the extending belt problem, the face belt is used and the material ploughed off the machine, which is not a very satisfactory arrangement. To make this system a success it is required to develop a belt conveyor that will retract or advance with the machine, the feeding of the material to be continuous and not interrupted by the movement of the conveyor.

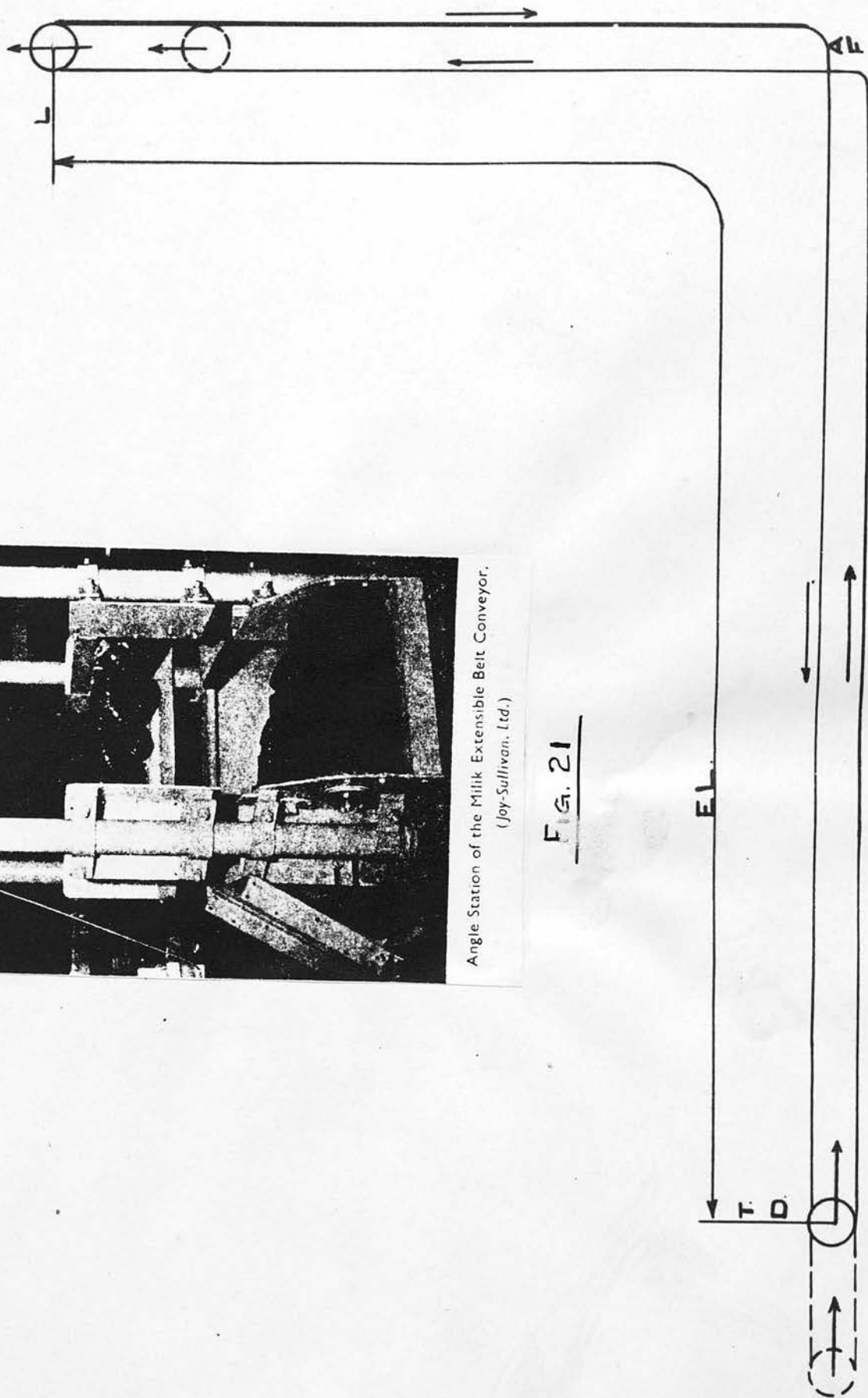
The solution of this problem may be in the design and development of the New Joy-Sullivan angle station conveyor. The main feature of this conveyor is the angle station, (Fig.21), a device enabling the belt to be turned through an angle of  $90^{\circ}$  or any other angle desired. The angle station employs rollers of special design to keep the belt in its correct path, and is self aligning. The belt may be deviated as





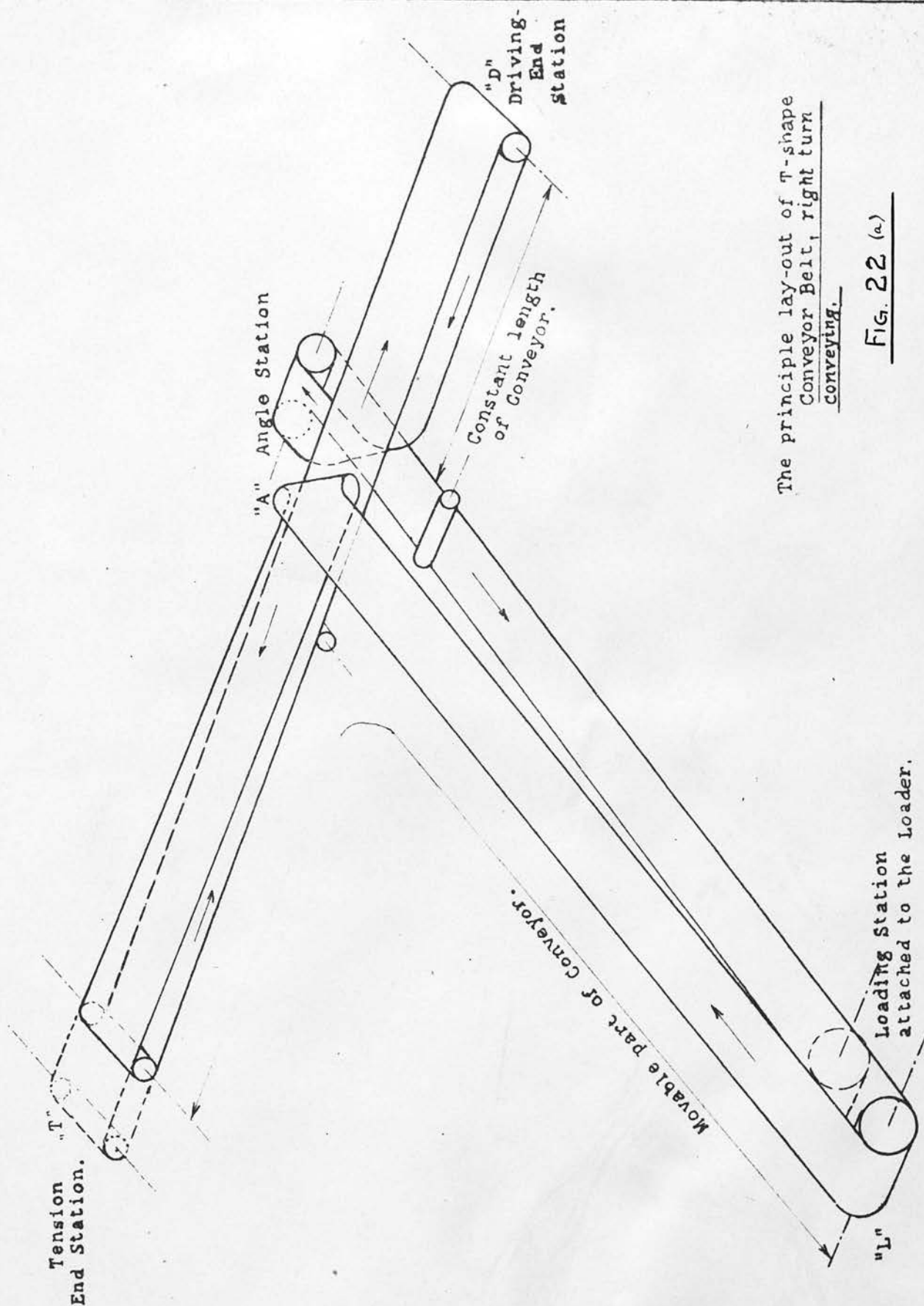
Angle Station of the Milik Extensible Belt Conveyor.  
(Joy-Sullivan, Ltd.)

FIG. 21



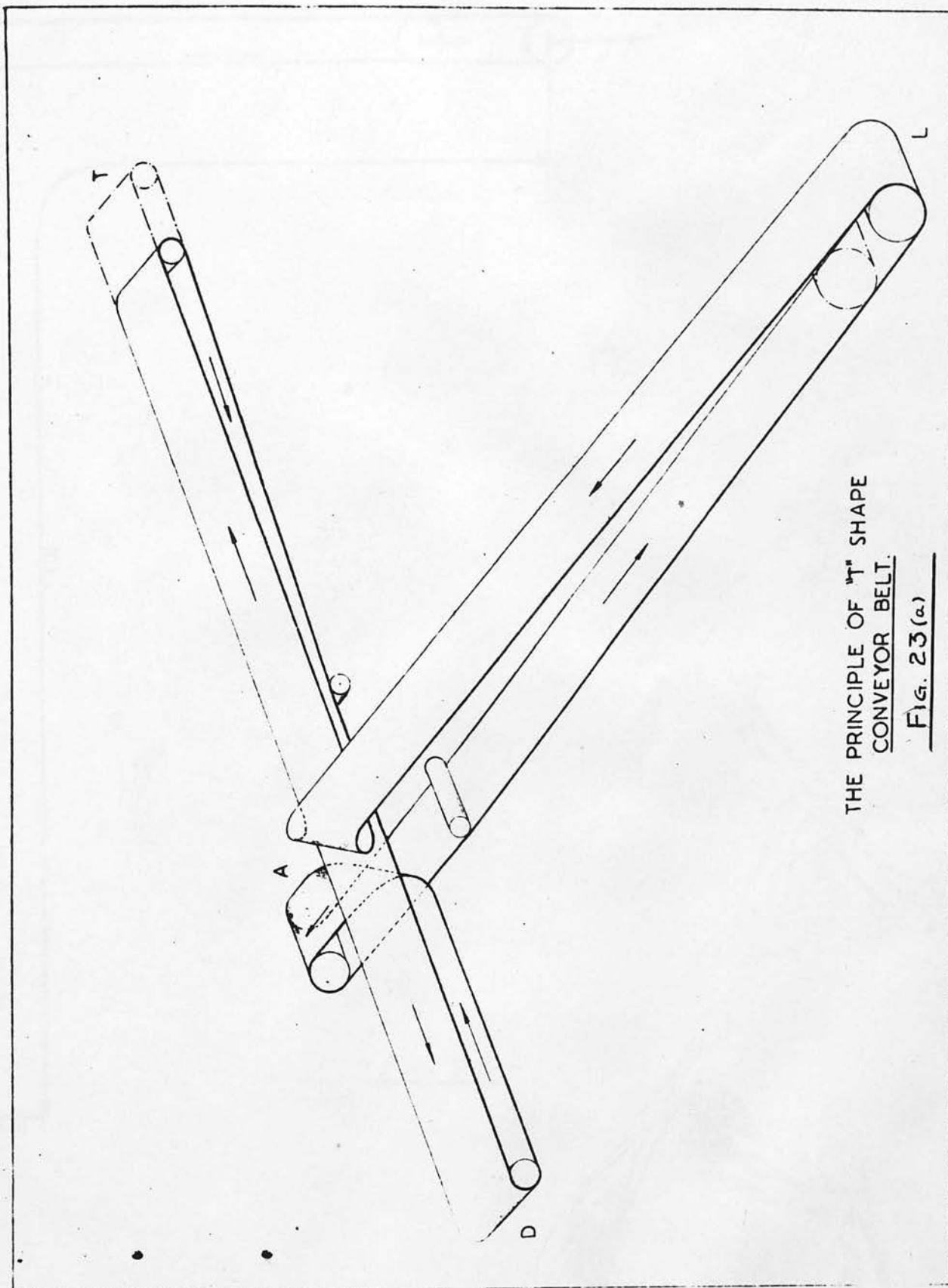
'L' - SHAPE EXTENSIBLE CONVEYOR LAYOUT - RIGHT TURN.

FIG. 22



The principle lay-out of T-shape  
Conveyor Belt, right turn  
Conveying.

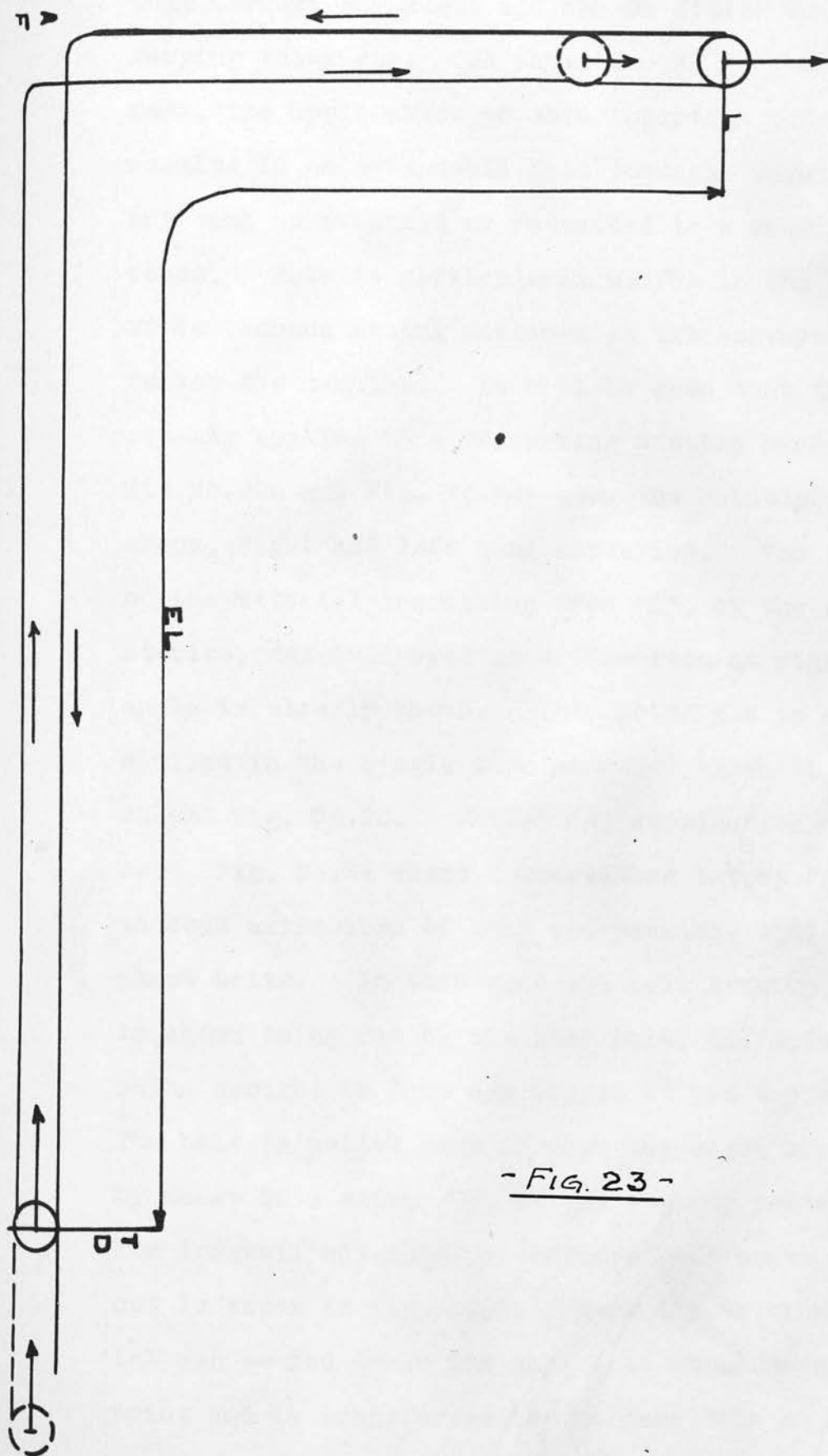
FIG. 22 (a)



THE PRINCIPLE OF "Y" SHAPE  
CONVEYOR BELT.

FIG. 23(a)

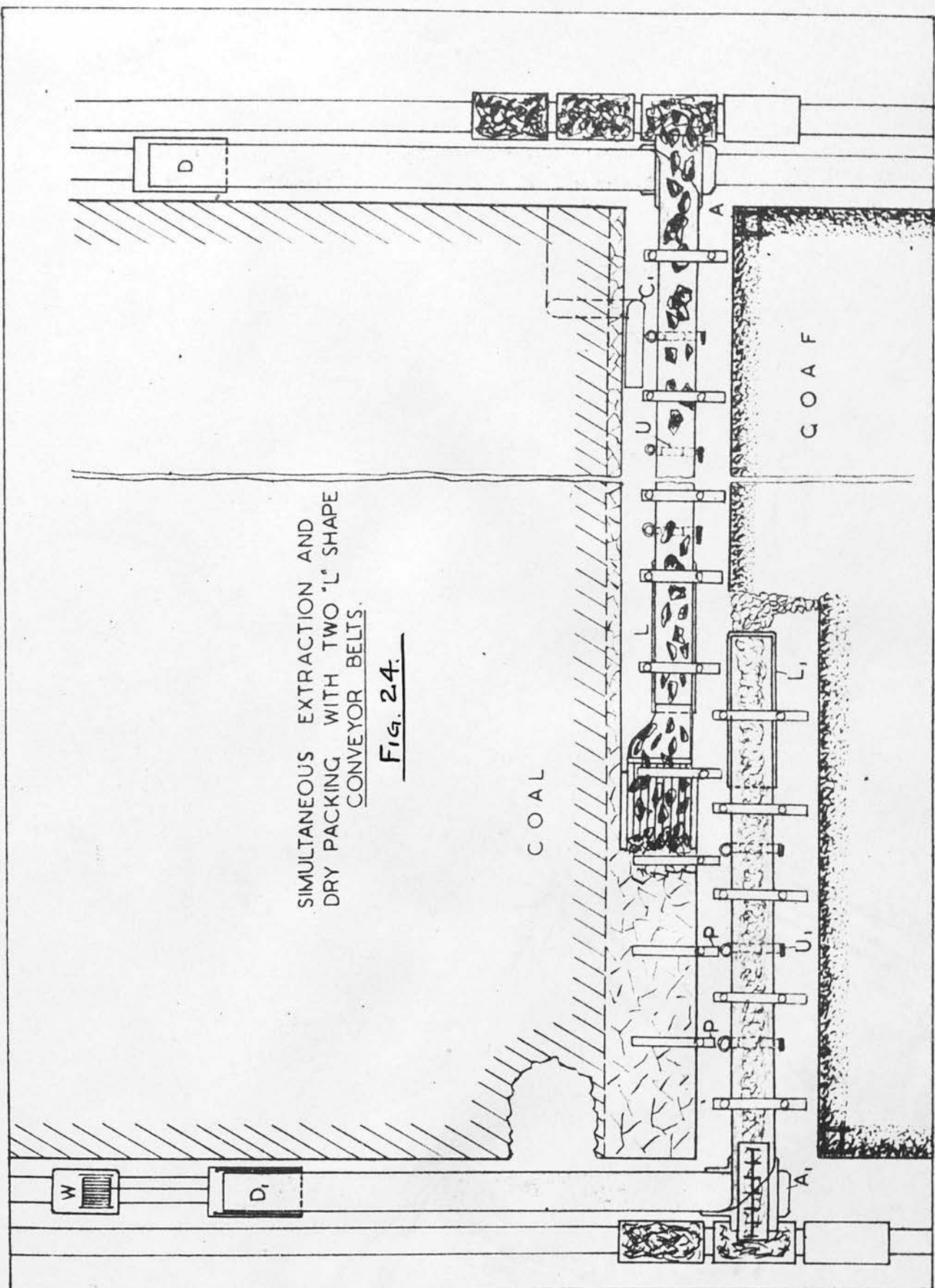
L-SHAPE EXTENSIBLE CONVEYOR LAYOUT - LEFT TURN



- FIG. 23 -



much as  $10^{\circ}$  either side and the angle station centralises the belt. The station may be made to turn the belt through any angle and can be fitted in seams of varying thickness. As shown in the following diagrams, the application of this important development results in an extendable belt conveyor which can at any time be extended or retracted to a required distance. This is particularly useful in the operation of continuous mining machines as the conveyor can follow the machine. It will be seen that this equally applies to a traversing stowing machine. Fig.No.22a and Fig. No.23a show the principle of T shape, right and left hand conveying. The transfer of the material travelling from "L", at the angle station, and delivered in a direction at right angle is clearly shown. This principle is also applied in the single turn conveyor shown in Fig.No. 22 and Fig. No.23. Whilst any combination can be used, Fig. No.24 shows a Retreating layout for simultaneous extraction of coal and packing, with two "L" shape belts. In this case the belt stowing machine is shown being fed by the goaf belt, the material being brought in tubs and tipped in the top road. The belt is pulled back through the angle station  $A_1$  by means of a winch "W", as the stowing proceeds. For longwall advancing a complete belt conveyor layout is shown in Fig.No.25. Here the stowing material can be fed on to the gate belt at a convenient point and is transferred to the face belt at angle station  $A_1$  and so to the belt stowing machine  $L_1$ .



SIMULTANEOUS EXTRACTION AND  
 DRY PACKING WITH TWO "L" SHAPE  
 CONVEYOR BELTS.

Fig. 24.

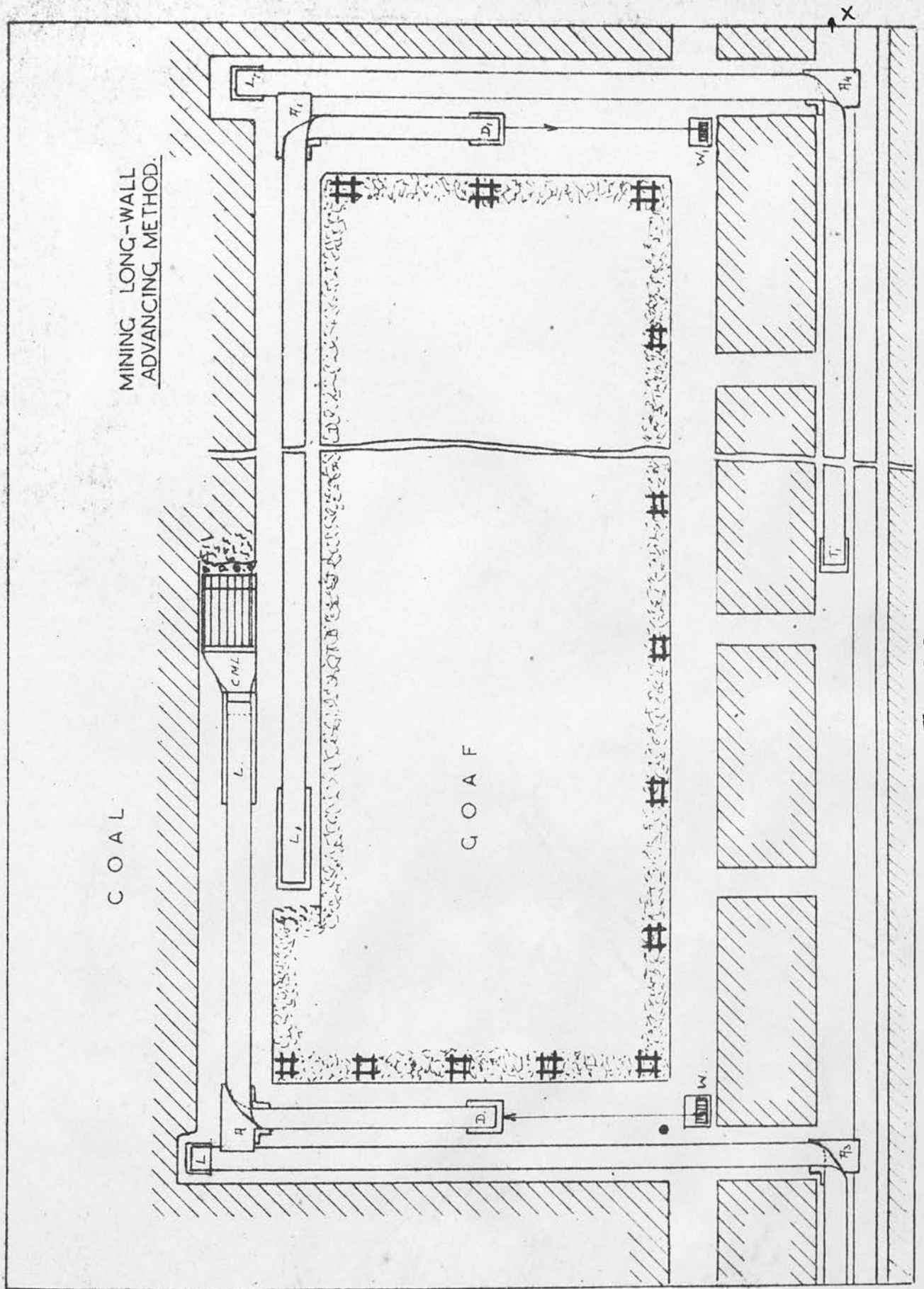


FIG. 25

The face belt is retracted by means of a winch  $W_1$  set in the supply road. The gate belt is also extendable to follow the face, and can be slightly in advance by means of a stable hole. If necessary the stowing material can be fed on to this belt on the main road, say, at point X and will be automatically transferred at angle station  $A_4$  and  $A_1$  to the stowing machine. The development of this conveyor offers new scope to the high speed belt stowing machine and the continued improvement in design and application is very desirable.



STOWING PIPES(A) INTRODUCTION(b) PIPE DESIGN

- (i) Material
- (ii) Dimensions
- (iii) Joints and Bends

(c) WEAR IN PIPES

- (i) General
- (ii) Flushing, Blasting and Impact Tests.
- (iii) Wear due to Friction and Impact

(d) PRACTICAL PERFORMANCE

- (i) Sand-Water Mixtures
- (ii) Air-Material Mixtures
- (iii) Pipe Life and Tonnages Handled.

(e) CONCLUSIONS

(a) INTRODUCTION

In hydraulic and pneumatic systems of stowing, the question of stowing pipes is of paramount importance, as the wear and replacement of these pipes constitute one of the heaviest operating charges. The lack of reliable practical data is to be regretted, but past results on hydraulic and recent figures on pneumatic stowing give some indication of the trend of design and of necessary fundamental conditions. The basic requirements are that the pipes should be light in weight to facilitate handling in the pit and should be so constructed to be able to withstand the abrasive action of the stowing material.

The durability of stowing pipes varies considerably with the following factors:-

- (1) Material used.
- (2) Type of stowing material flushed.
- (3) The water or air blast velocity.
- (4) The ratio of stowing material to air/water.
- (5) the length of the pipe lines.
- (6) The course of the pipe lines (bends, etc)
- (7) Standard of installation (joints, etc)
- (8) Attention to maintenance.

The stowing performance achieved in practice until the pipes are unserviceable may vary from 10,000--100,000 cu.yds. filling material, depending upon the above factor promoting wear.

If this mining system is to develop, this very marked variation in durability of stowing pipes, coupled with the expenditure involved in replacement, warrants the necessity for careful investigations regarding the factors determining the wear-resistant properties of pipes used for this work.

(b) PIPE DESIGN(i) Material

Tests have been made on pipes constructed from the following materials:-

Wood (Flumes)  
Terra Cotta  
Cast Iron  
Drawn steel tubes  
Lined Pipes

Each material can be used with advantage in special circumstances and conditions, but the general trend is to use special drawn steel tubes. Certain lengths and bends may come under the lined pipe category. There is, however, still a large field for the use of cast iron pipes - especially in hydraulic stowing - and use in stationary situations, levels, shafts, etc., would appear to have definite advantages. Amongst the advantages that can be claimed are cheapness, good wearing properties against abrasive materials, strength, and eventual scrap value.

There is no doubt, however, that drawn steel pipes are best for face use, due mainly to the lighter weight and therefore ease of handling. These pipes are of carbon steel and may be of the following strengths:-

Specification strength:-

37.11	50.11	60.11.	70.11
-------	-------	--------	-------

Ultimate tensile strength in Kgms.mm<sup>2</sup>:-

37/45	50/60	60/70	70/85
-------	-------	-------	-------

Ultimate tensile strength in Tons per sq.in:-

23.5/28.6	31.8/38.1	38.1/44.4	44.4/54
-----------	-----------	-----------	---------

(ii) Dimensions

The overall dimensions of stowing pipes vary considerably and no hard fixed rule can be stated. An important practical feature is that with pipes of suitable dimensions, mechanised stowing as regards hydraulic and pneumatic systems, can be practised in thin seams.

Level and other semi-permanent pipe lines can be made up of pipes of larger standard sizes. In cases where shaft installations are necessary, it is of considerable advantage to have the shaft pipes in as long lengths as possible to cut down the number of joints necessary.

For hydraulic stowing the writer found it advantageous to install cast iron shaft pipes in 9ft. lengths, 5.5" with 1" walls, with straight machined flanges containing six  $\frac{3}{4}$ " bolt holes, the weight of these pipes being approx. 280 lbs. The level pipes were of similar dimensions, but the walls were reduced to  $\frac{3}{4}$ " thickness.

In pneumatic stowing the common size of carbon steel pipe is in standard lengths of 10 ft. having a wall thickness of  $\frac{3}{16}$ " and a diameter of 6". Continental practice is to use drawn steel tubes 3 metres long with 5 mm. thick walls and a diameter of 160 mm., weighing approx. 230 lbs. with couplings. For level pipes it is recommended that the wall thickness be increased to  $\frac{3}{8}$ " and the length increased to 12 ft.

Face pipes, having to be man-handled every



stowing shift, must be of light construction and of a length suitable for the thickness of the seam. In fairly thick seams drawn carbon steel tubes in 10 ft. lengths, 3/16" walls, 6" diameter, are suitable, whilst in thin seams, pipes of 5'6" are easier handled. In some instances, face pipes of a smaller diameter 5" and 4" have been used with considerable saving, but the reduction in diameter greatly increases the risk of pipe blockage. This development is closely bound up with stowing material size. This practice of diameter reduction does give tangible practical results in a more constant and steady flow of material from the pipe mouth in hydraulic stowing and a reduced air consumption per ton stowed in pneumatic stowing.

Circular pipes have been found to be entirely suitable for stowing operations. Oval pipes have been tried but difficulties in manufacture and extra cost outweigh any advantage gained. Outside thickening of the pipe has been found advantageous on bends and special pipes where there is a change of direction or turbulent flow.

The consideration of diameter is of fundamental importance in the design of a pipe unit. In fixing this factor, the criterion is the quantity of material to be conveyed in a given time and its maximum size. Modern machine mining will demand maximum tonnage per hour consistent with pipe wear. This figure is likely to be in the region of 200 tons per hour.

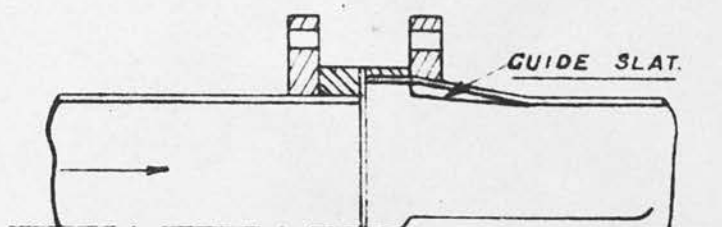
(iii) Joints and BendsJoints

Where pipes are used for mechanised stowage the selection of pipe joints fall into two classes:-

- (I) Joints for permanent or semi-permanent pipe lines;
- (II) Quick release joints for face pipes;

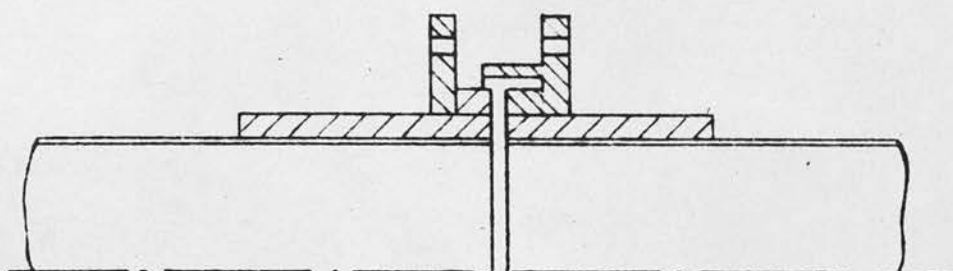
(I) In this category, a simple standard joint is to be preferred either screwed socket or screwed and welded flange for drawn steel tubes or integral cast machined flange joint for cast iron ranges. Where flanges are used either 4 or 6 bolt holes can be provided, depending upon the type of joint packing to be used. For water it was found in practice that a 6 hole flange with asbestos jointings gave good service in shaft ranges, whilst a 4 hole flange was suitable for level pipes. This type of packing was found to be more economical than rubberised joints. A simple jointing material was used in hydraulic operations, the packing consisting of flat circular steel rings wrapped round with raw jute. For main air lines the ordinary commercial air jointing rings have been found quite suitable.

In order to reduce wear caused by impact near pipe joints it is necessary to avoid projecting edges of unevenly-laid pipes. The following sketches give diagrammatic plans of typical joint arrangements to ensure in-line pipes (Figs.26-31).



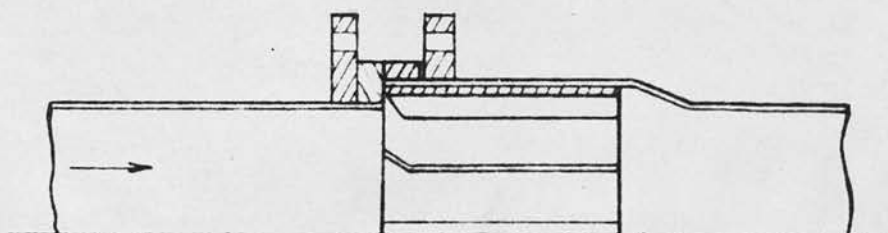
ENLARGED PIPE WITH PRESSED IN SLAT.

FIG. 26.



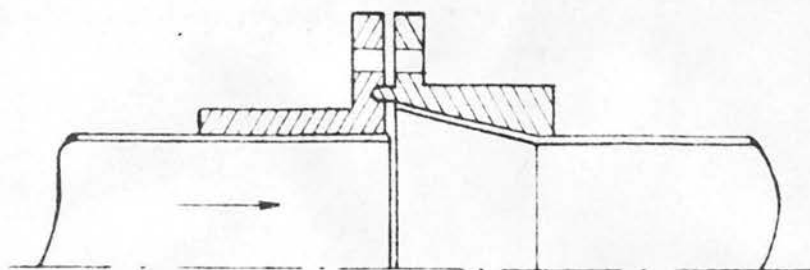
PIPE ALIGNMENT BY MEANS OF  
A WITHDRAWABLE FLANGE.

FIG. 27.



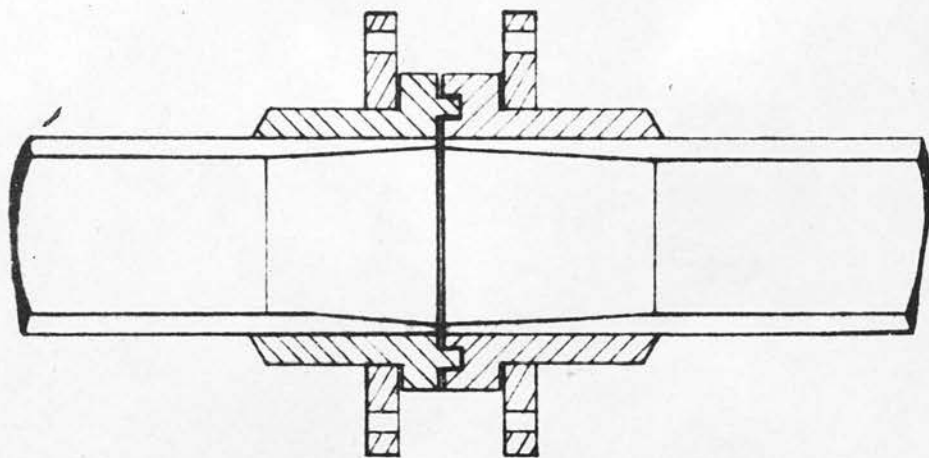
CYLINDRICALLY ENLARGED IN-GOING PIPE WITH  
PRESSED-IN ANTI-WEAR RING.

FIG. 28.



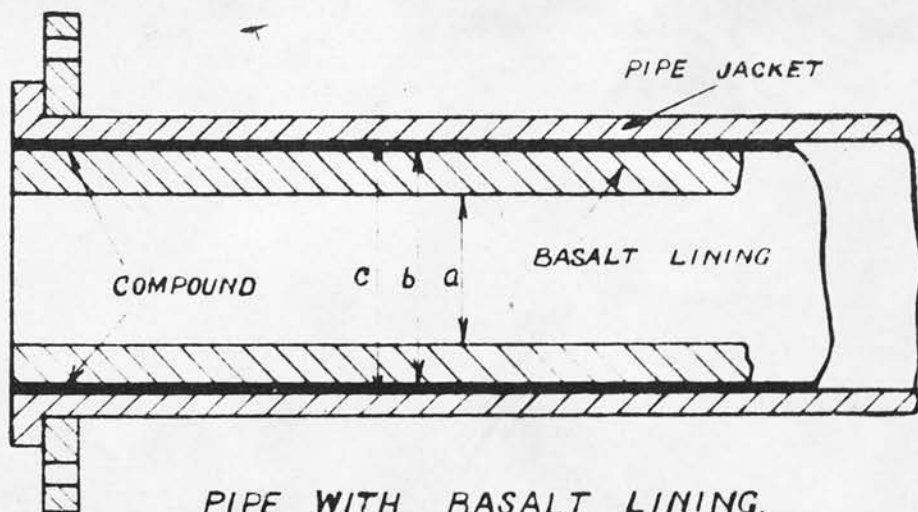
CENTRAL ALIGNMENT OF PIPES AND CONICAL  
ENLARGEMENT.

FIG. 29.



CENTRALLY ALIGNED BY GROOVE AND RING  
AND ENDS CONICALLY BORED OUT.

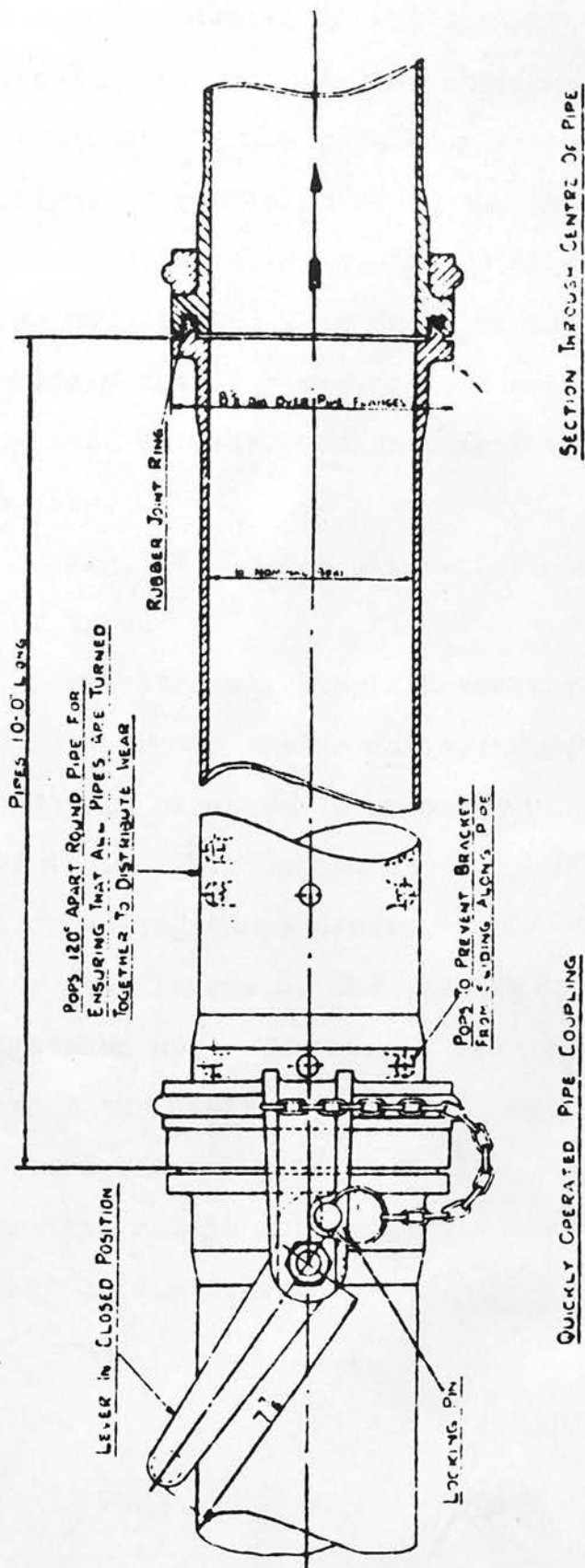
FIG. 30.



PIPE WITH BASALT LINING.

FIG. 31.





## BLASTOWER

### GENERAL ARRANGEMENT OF STOWAGE PIPES

Fig. 32.

(11) The type of joint used on face pipes must be quickly attachable and detachable. This is especially important when continuous stowing is practised. There have been several **types** designed for this purpose, the most popular being the spigoted rubber-sealed joint with the Hammacher type coupling. This joint is not, however, entirely satisfactory, as the cam head and fulcrum pin tend to wear quicker than the pipe and leakage results.

Fig. 32 shows a self-explanatory joint of this type.

An extremely simple detaching arrangement was devised by the writer for hydraulic face pipe lines to enable pipes to be released without stopping the stowing. This consisted of a heavy U clamp with a quick-acting thumb screw. This clamp was fitted over the flanges of the pipe to be released and tightened up. The flange bolts were then removed and at the desired time the clamp unscrewed, which allowed the end pipe to fall. There is no reason why this simple arrangement should not prove efficient in air lines.

Bends

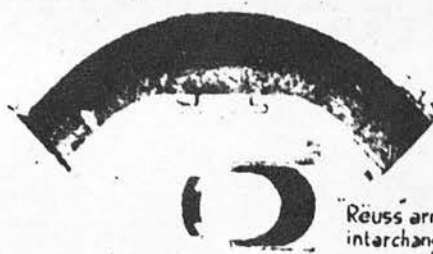
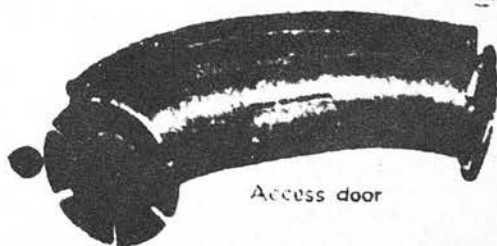
It is an axiom in hydraulic and pneumatic stowing that there should be good alignment of the pipe range and curves should be avoided if possible. At each bend there is a loss of efficiency, therefore the number should be kept as low as possible. The bends sustain much more wear than straight lengths, due to the fact that the material in the pipe has to change direction at speed. As there is a drop in velocity at each bend, the material requires some length of straight pipe to regain speed, otherwise bends placed in too close proximity may cause blockages. Bends should be constructed as to be quickly accessible for cleaning and should be provided with re-newable liners.

The radius of curvature should be as large as possible, if necessary the pipe bends being made up in sections, a good arrangement being that the sections be made up in  $7\frac{1}{2}^{\circ}$ ,  $15^{\circ}$ , and  $30^{\circ}$  pieces, so that adjustments of the angle of curve can easily be made.

In South Wales, 18" radius cast steel right-angled bends with replaceable cast iron liners are used, and it is claimed that the lines last for 12,000 cu.yds. of washery shale and 2,500 cu.yds. of crushed rubbish. For bends other than a right angle chrome iron cast eccentric 1" thick on the outside and  $\frac{1}{2}$ " on the inside have been found satisfactory.

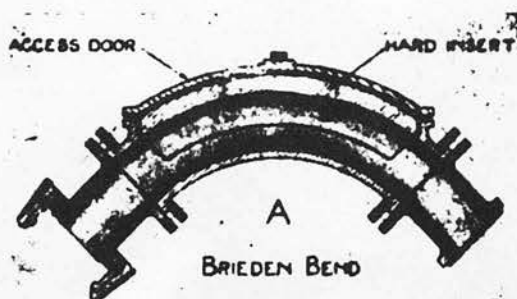
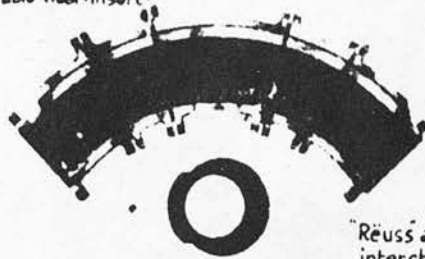
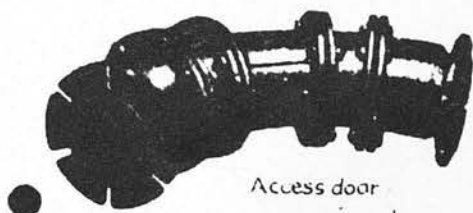
"Rëuss" special 90° bends in the main connecting the road with the face.

Hartman cast-steel  
interchangeable wear insert



"Rëuss" adjustable segments for 30°, 45°, 60°, 75° and 90° bends.

Hartman cast-steel  
interchangeable wear insert



Adjustable section for bends  
from 8° to 20°



Adjustable connecting segments  
for 30°, 45°, 60°, 75° and 90° bends

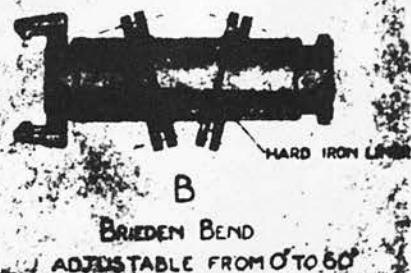


FIG. 33.

Various types of bend.



Various classes of material have been tried for inserts, including granite, rubber and basalt, but have not been entirely satisfactory. Bends made of high silicon manganese iron have proved to be the most economical. More complicated bends using air jets have not yet reached practical use. A simple easy renewable bend in cheap material has much to commend it.

(Insert prints of different bends) (Fig.33).

(c) WEAR IN PIPES

(i) General

In connection with this important problem, considerable work has been done in Germany - mainly by Wellinger and Brockstedt at Stuttgart, who determined the wear-resistance of various materials intended for manufacture of pneumatic and hydraulic stowing pipes, and also on the influence of the laying of the pipes for pneumatic stowing plants. The test results not only elucidated how wear is influenced by the nature of the pipe material and the manner in which the pipe lines are laid, but also brought proof in figures of the importance of other factors, the nature of which was already known from practical experience.

It therefore seems advisable to summarise all information available from laboratory work and practical experience pertaining to this aspect of the subject.

In order to make the essential test results more comprehensible, only some of the outstanding test values have been selected and have been represented by graphs. The values for wear are converted to express the decrease in thickness of the pipe wall per working hour, and by drawing the curves for wear in comparative scales, it brings the hydraulic and pneumatic test results into relationship with each other and makes possible the comparison in wear per mm. hour directly with wear of pipes in service.

(ii) Results of Flushing, Blasting and Impact Tests\*

Flushing Test (Fig. 34). In this test, a steel cylinder 10.4 mm. outside diameter and 100 mm. long was rotated in a vessel containing a mixture of sand and water, the sand being  $< 5$  mm. diameter, the ratio of sand to water being 2 : 1. The peripheral velocity in metres per second is plotted against the wear in 1/1000 mm. per hour.

It was found that abrasion increases considerably with increasing flow of velocities, and is much greater with soft steel than with harder varieties. Thus, for instance,  $V = 16$  m/sec. the wear of pipes made of steel with U.T.S. 23.5/28.6 tons sq. in. is 1.31 times that of pipes of U.T.S. 31.8/38.1 tons sq. in. and 5.5 times that of hardened steel (U.T.S. 44.4/54 tons sq. in.) The choice of a suitable steel quality is the predominant factor in determining the durability of stowing pipes. The results show that the resistance of steel to frictional abrasion is approx. proportionate to its hardness.

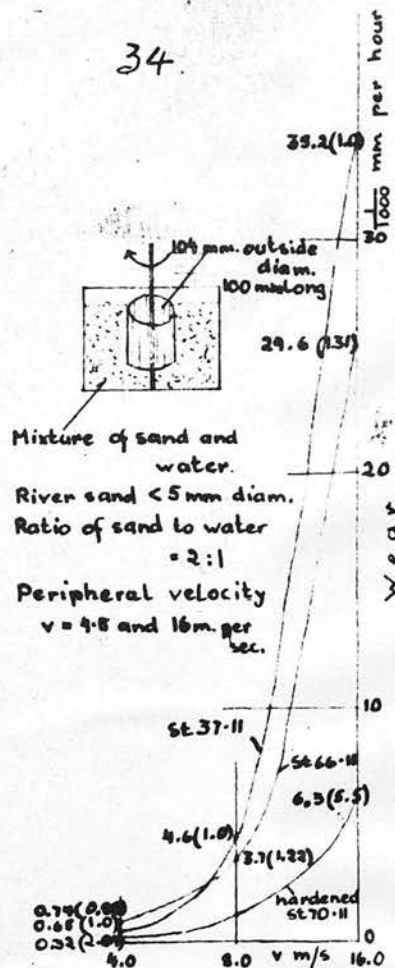
A surprising fact is that the increase in abrasion is greater than that which would correspond with the squares of the velocities; e.g. take steel U.T.S. 23.5/28.6 tons sq. in.  $= \frac{35.2}{0.65} : \frac{16^2}{4^2} = \frac{54.2}{16} = 3.4$  times

and taking hardened steel (U.T.S. 44.4/54 tons sq. in.)

$$\frac{6.3}{0.3^2} : \frac{16^2}{4^2} = \frac{19.7}{16} = 1.23 \text{ times greater}$$

\* GUTACKER. BONN. "GLUCKAUF" Vol. 81-84. 1948.

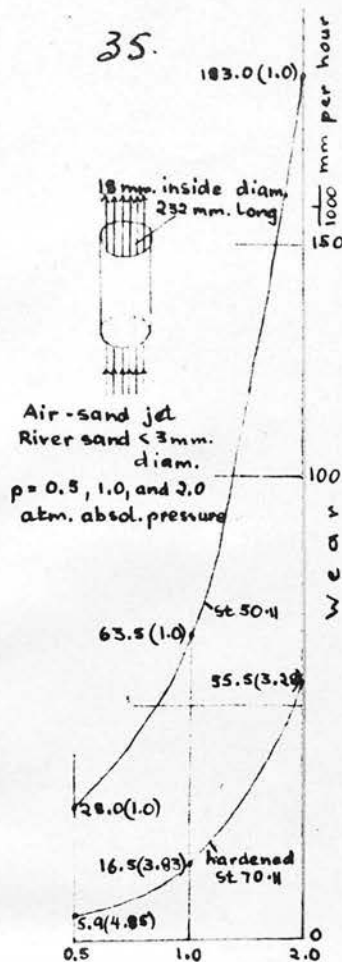
34.



Peripheral Velocity  
Fig. 34

Test Arrangement 1  
Pipe section rotating  
in mixture of sand  
and water.

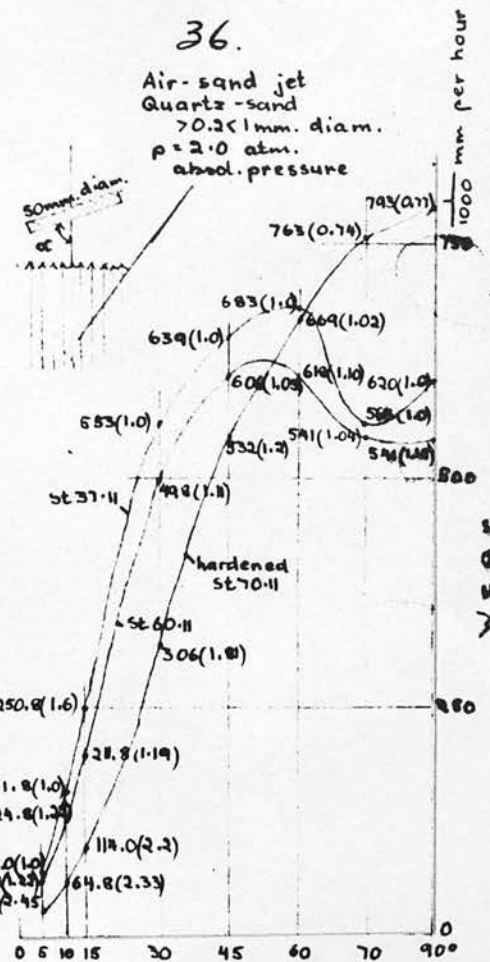
35.



Air-Blast Pressure  
Fig. 35.

Test Arrangement 2  
Sand blasting  
through pipe.

36.

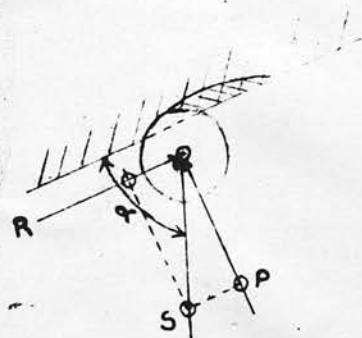


Jet Angle  $\alpha$   
Fig. 36.

Test Arrangement 3  
Sand-blast jet directed  
against plates at  
different angles.

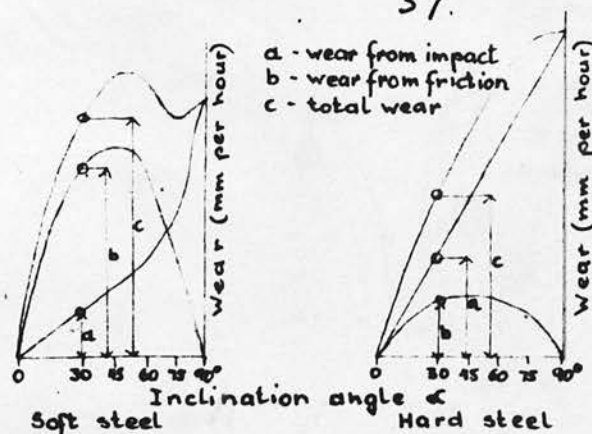
In brackets the wear-resistance, i.e. the ratio of wear of St 37-II to wear of St 60-II and hardened St 70-II.

### Figs 34-37 Tests for Determining the Wear-Resistance of Various Materials.



Conjoint action of  
wear due to impact and  
friction with jet striking  
plate at an angle.

37.



Wear due to impact  
and friction of a steel plate  
against which a sand blast is  
projected at an angle  $\alpha$ .

Fig. 37.



Degrees of wear were also recorded for different types of stowing material. The grain size of the material had no important influence on wear, although smaller grains led to a slightly higher abrasion. It is suggested that a material suitable for hydraulic stowing should contain at least 50% grains of more than 6 mm. diameter. The flushing velocity is dependent upon the nature of the stowing material and the size of its particles upon the amount of flushing water and the length and slope of the pipe lines. As there is a marked increase in abrasion due to increased flushing velocities, the slope should not exceed that which is necessary to prevent clogging.

The amount of water added does not have a marked influence on wear. At 8 m/sec. velocity, and addition of six parts water, the abrasion with hard steel (U.T.S. 44.4/54 tons sq.in.) was approx.  $\frac{3}{4}$  of that when only two parts were added.

As in the impact test, even a slight inclination of the flushing stream against the pipe walls led to a considerable increase in abrasion.

As most of the factors having an influence on wear are to be regarded as unavoidable attributes of the prevailing local conditions, the only effective means of reducing wear and thereby substantially reducing expenses for wastage, labour and running costs, will be to choose a suitable pipe material for the stowage to be handled.

Blasting Test (Fig. 35) This test consisted of blowing sand at 0.5, 1.0 and 2.0 atmospheres absolute pressure through a pipe 18 mm. inside diameter and 232 mm. long. The river sand size was  $< 3$  mm. The graph shows air blast pressure plotted against wear.

The results showed a considerable increase in wear with rising working pressures, i.e. increasing velocities. In contrast to the results of hydraulic tests, the increase in wear here is inversely proportional to the hardness of the material. In spite of this, a marked superiority of hardened steel was found. Abrasion of steel (U.T.S. 31.8/38.1 tons sq.in.) at 2 atmos. absolute pressure was 3.28 times and, at a pressure of 0.5 atmos. it was 4.85 times that of hardened steel (U.T.S. 44.4/54 tons sq.in.)

In contrast to the results of the hydraulic tests, the increase in wear in the pneumatic tests is lower than that corresponding to the square of the blast pressure, i.e. the square of the velocities.

For instance with steel (U.T.S. 31.8/38.1 tons sq.in.)

$$= \frac{183.3}{28.0} : \frac{2.0^2}{0.5^2} = \frac{6.54}{16} = 0.41$$

and hardened steel U.T.S. 44.4/54 tons p. sq. in.

$$= \frac{55.5}{5.9} : \frac{2.0^2}{0.5^2} = \frac{9.41}{16} = 0.59$$

Whereas in the hydraulic system the ratio of abrasion of the softer steel qualities to the harder types increases with increasing flushing velocities, in the blasting process this ratio increases with

decreasing pressures, i.e. with diminishing blast velocities.

Impact Test (Fig. 36) An air-sand jet at 2.0 atmospheres absolute pressure was directed against steel plates lying at different angles. The sand was quartz sand  $> 0.2 < 1$  m. diam. The graph shows the jet angle plotted against plate wear in 1/1000 mm. per hour. W. by ?

These tests showed that the wear of all steel qualities first increases uniformly up to a jet angle of  $45^\circ$  and  $60^\circ$  respectively, the harder steel types showing superior wear-resisting properties to those of softer types up to an angle of  $50^\circ$ . The wear resistance ratio of soft steel (U.T.S. 23.5/28.6 tons sq. in.) to hardened steel (U.T.S. 44.4/54 tons sq. in.) at an angle of  $30^\circ$  ~~is~~ is 1:1.81. At an angle of  $60^\circ$ , however, an inversion of the ratio takes place, the wear of the softer steel suddenly diminishing, whilst that of the hardened steel still increases and even surpasses that of the softer steel. In other words, whereas the hardened steel is vastly superior to softer steel in the flushing and blasting wear and also when projected at angles up to  $45^\circ$  against walls, the absolute wear of hardened steel exceeds that of the softer steel at jet angles of  $60^\circ$  to  $90^\circ$ , though only moderately. This is an important point in the abrasive influences on pipe bends. High grade alloy steel does not show any marked superiority in wear resistance to warrant the increased expense.

(iii) Wear due to Friction and Impact

The variant behaviour of very hard types of steel and the softer qualities in the hydraulic tests with obviously parallel flow on the one hand, and in the pneumatic tests with turbulent currents on the other hand, and furthermore, the remarkable course of the curves of wear in the impact test in the range of a jet with inclination of from  $50^{\circ}$ - $60^{\circ}$ , gives rise to the supposition that these phenomena of wear are due to two abrasive processes distinctly different in their reactions.

It is helpful for understanding the nature of these abrasive phenomena to imagine the kinetic energy  $\frac{1}{2} m v^2$  of a grain of sand of mass  $m$  and impinging on a steel plate at an angle  $\alpha$ ; is converted into energy of deformation. (Fig. 37). On resolving the energy vector  $S$  into components  $R$  and  $P$ , parallel and perpendicular to the plate, we find that the deformation effects caused by each of the components have widely differing characteristics.

The energy component  $P$  causes the grain of sand to impinge on the steel plate and endeavours to press the granule into the surface of the plate. According to the value of the co-efficients of elasticity and plasticity of the grain of sand and of the steel plate, the energy component  $P$  will dislocate particles of matter from the structure of the surface of the steel plate, similar to the phenomena of cavi-



tation, and so will cause air "impact or shock wear". If  $\alpha = 0$ ,  $R = 0$ , and with  $\alpha = 90^\circ$ ,  $P$  attains a maximum value. The energy component  $R$ , on the other hand, has the tendency to shear or hone particles out of the steel plate surface, and so it causes a frictional or shearing abrasion. But the component  $R$  can only succeed in doing this if the component  $P$  presses the grain of sand against the steel plate. From this it follows that whereas impact wear is only a function of the energy component  $P$ , the frictional abrasion is a function of both the energy components  $P$  and  $R$ . So if  $\alpha = 0$ , the frictional abrasion must equal zero, because  $P = 0$ , although in this case  $R$  attains the maximum. If  $\alpha = 90^\circ$ ,  $P$  attains the maximum value, but then  $R = 0$ , therefore the frictional wear in this case will equal zero. The maximum value of frictional wear is reached at an inclination of  $45^\circ$ . It is only in the range where the effect of percussive wear is predominant that we encounter a more favourable behaviour of the softer steel types.

The deformation reactions leading to wear do not, however, become effective one after the other, but act simultaneously. In reality, the deformation forces are considerably more intricate because they are dependant upon a great number of factors. However, the above reasoning serves to convey an idea of the complicated phenomena of wear and gives a possible

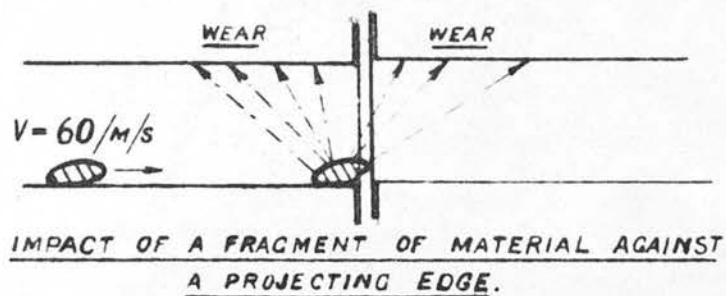


FIG. 38.

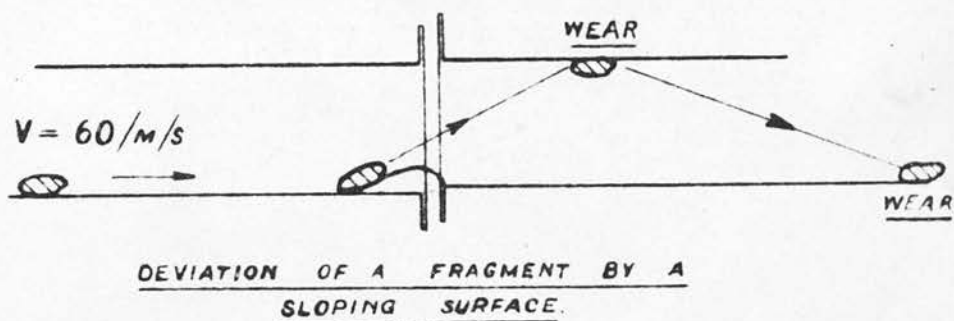
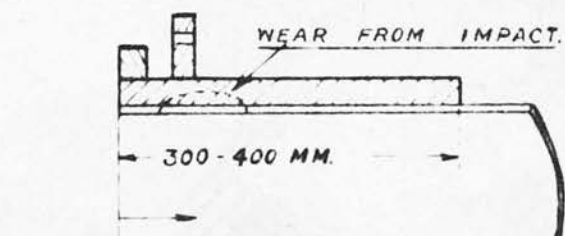


FIG. 39.



WELDED ON SLEEVE.

FIG. 39 a

explanation of wear relating to combined percussive and frictional forces.

As a practical instance of the above, it is observed that both hydraulic and pneumatic stowing pipes often show excessive wear and local pitting near the joints. This is due to impact of the fragment striking against the sharp edge of a badly-laid pipe. <sup>In Fig. 38</sup> ^ The arrows show the direction of the fragment hitting the pipe round the joint. With speeds up to 200 ft. per second, the force in such cases is considerable. After a time the destructive processes can also take place as shown in Fig. 39, which shows the deviation of a piece of stowing material by a sloping surface resulting from accumulation of small material against a projecting flange. By both processes hollows are first formed at the places of impact and eventually cause holes in the pipes. The wear caused by impact is far more rapid than that caused by friction on the bottom of the pipes. (Fig. 39(a)).

(d) PRACTICAL PERFORMANCE(i) Sand-water Mixtures

The amount of sand that can be flushed in a pipe in a given time depends upon several factors:-

- (i) Length of the pipe lines.
- (ii) Course of the pipe lines (nos. of bends)
- (iii) Diameter of pipe line.
- (iv) Water velocity.
- (v) Ratio of water to sand.

In practice the limiting factors are Nos. i, ii, and iii. As the pipe line extends, so the quantity flushed per hour falls until a point is reached where stowing becomes uneconomic. This limiting distance falls rapidly with pipe diameter and with adverse grade. The ratio of sand to water also varies considerably; for short lengths of pipe the ratio may be 1:1. In fairly long pipe lines, (3,000-5,000 ft.) an average ratio with a 6" diameter pipe and adequate fall would be in the region of 1 : 3-5.

Careful measurements of pressure drop on a 5,000 ft. length of straight  $3\frac{1}{8}$ " diameter pipe line have shown that within the limits of experimental error the friction factor remains constant for pulps ranging from 40 - 60% solids. That is, increase of pulp density raises the line pressures only by an amount equal to the increased static head. This result is contrary to expectation, but may be explained by reference to Reynolds' formula for turbulent flow

16th 67?



in pipes:-

$$h = \frac{(Kv^n v^{2-n})}{d^{3-n}} \times l$$

where h is loss of head in feet of mixture in length  
l;

K = a constant

v = velocity

d = diameter of pipe

V = M/P where M equals viscosity, P = density

Apparently in some pulps increase in density is accompanied by a proportional increase in viscosity such as to keep the quantity M/P constant. Hence for similar velocity in the same pipe loss of head is independent of density. This result probably holds good for any pulp in which the solids are homogeneous and free from flat particles. Such particles would, of course, cause an increase in viscosity at a greater rate than that in density.

#### (ii) Air-Material Ratios

Here again the carrying power of compressed air is governed by similar factors as enumerated under sub section (a), in addition the shape and size of the stowing material having a more important bearing on the performance. The air pressure required and the air consumption per cu.yd. stowed are also dependent on similar factors. It is estimated by various authorities that the pressures necessary in practical performance varies between 25 and 50 lbs. per sq.in., the total air consumption depending upon the machine features ranging from 40 - 120 cu.ft. air per cu.ft.

material stowed.

With materials normally used for pneumatic stowing an air velocity of about 200 ft.per sec. is required to carry and discharge at sufficient velocity to build a tight pack. It is probable that some of the material in the pipe is completely air-borne and travels at a near relative speed whilst the heavier parts will be pushed along the pipe bottom as evidenced by wear. Some German engineers estimate that the material lags behind the air velocity by one quarter to one half the theoretical velocity. In general the aim should be to stow with the minimum quantity of air and this can only be done in practice by trial and error. Suffice it to say that the machine should be designed to be capable of handling the maximum quantity likely to be handled and provision made for a less quantity to be stowed economically.

### (iii) Pipe Life and Tonnages Handled

The life of piping, which varies considerably, depends upon the following main factors:-

1. Constructional material of pipe
2. Quality of stowing material
3. Layout and maintenance of pipe lines

The air blast or water velocity and the ratio of the transporting medium to filling material also play important parts in pipe wear. Owing to these variations it is not possible to forecast accurately the life of stowing pipes, although in set

circumstances this can be done with fair approximation.

Careful maintenance and accurate recording of tonnage handled will enable the pipe life to be extended considerably. With steel pipes pop marks on the outside or on cast iron pipes numbers cast on the flanges, enables a pipe line to be turned, so that wear can be evenly distributed. Common practice is to turn the pipes 120°.

In general the life of pipes may range from 10,000 tons for hard abrasive material to 100,000 tons for softer debris.

A few specific cases are given:-

TABLE A

District	Pipe Diam.	Material	Thickness	Tonnage
S. Yorks	6"	High Carbon Steel	3/16"	30,000 No appreciable wear
S. Wales	6"	"	"	35,000 tons shale wear 1/16 ins.
Scotland	6"	"	"	90,000 tons washery refuse and shale.
India	5.5"	Cast Iron	$\frac{3}{4}$ "	240,000 tons sand
Belgium	6"	Polished Steel	80-90 lbs p. yd.	130,000 cubic yards
Germany	150 mm.	38/40 U.T.S.	5. mm.	45,000 cubic metres
Germany	150 mm.	60/70	5 mm.	67,000 cubic metres.
Germany	150 mm.	Basalt Lined Steel		180,000 cu. metres.

better all in line for comparison

Table B gives particulars of tonnage figures.

recorded by the writer at a large mine in Bengal. The first column gives the name and location of the pipe range, the second column gives the length of range fitted together, the third indicates the mark on the flange, i.e. the degrees turned, Mark 1 installed position, Mark 2 turned 120° usually turned at 80-100,000 tons, Mark 3 turned 240° from original position and run to destruction. The four last lines show recorded tonnages through the stowing shaft pipes, 9 ft. lengths 5.5" diameter with 1" thick walls. These pipes were not turned as they are fitted vertically in the shaft.

TABLE B

Range	Length in ft.	Mark	Tons flushed up to Sept. 1945
<u>13 Dip range serving 3R</u>			
3 Pit to 16 level	780	Mark 2	310,000
16 level to 14 level	200	" 2	298,500
14 level to 10 level	380	" 2	250,500
10 level to 7 level	180	" 1	92,000
<u>13 Dip range serving 8 Dip</u>			
3 Pit to 15 level	880	" 3	308,000
15 level to 1 cross-cut	1680	" 3	255,000
1 cross-cut to 5 level	850	" 3	360,000
<u>2 Pit to 8 Dip East</u>			
Pit bottom to 3 Dip	420	" 3	319,000
3 Dip to 4 Dip	280	" 2	225,000
4 Dip to 8 Dip	500	" 3	224,000
8 Dip top to 3 cross-cut	600	" 1	75,000
Along 3 cross-cut	1400	" 2	207,000
Along 1 C cross-cut	1600	" 2	145,000
<u>3 Dip range</u>	1850	" 1	14,000
<u>2 Pit East shaft range</u>	1400	) Shaft ranges	148,000
<u>2 pit West shaft range</u>	1400		17,713
<u>3 Pit shaft range to 3R</u>	1000	) not turned	120,000
<u>3 Pit shaft range to 8 Dip West</u>	1000		185,000



(e) CONCLUSIONS

Tests and experience gained in practical service seem to prove that hardened steel pipes, especially the double wall material with a hardened wear resistant lining of hardened 70/85 tons sq.ins. U.T.S. is superior to softer types of steel in cases where frictional abrasive forces are predominant. The lining of stowing pipes in levels with basalt is undoubtedly advantageous, but further investigation is required especially in manufacturing problems. In certain specific instances especially in hydraulic stowing there is much to be said for suitably designed cast iron pipes. The question of chilled fine grain cast iron pipes for stationary locations should receive attention as there may be favourable economic balance both in hydraulic and pneumatic systems.

SURVEY OF STOWING PLANT

(a) HYDRAULIC

- (i) History and Development.
- (ii) Modern Plant and Practice.

(b) PNEUMATIC

- (i) History and Development.
- (ii) Modern Plant.
- (iii) Power Consumption.

(c) MECHANICAL

- (i) History and Development.
- (ii) Modern Plant.
- (iii) Theoretical Considerations in  
New Designs.

SURVEY OF STOWING PLANT(a) HYDRAULIC(i) History and Development

This method of stowage is said to have originated in the accidental discovery of the almost perfect filling obtained in a mine in Pennsylvania where the effluent from a coal washer was turned into old workings which became silted up. It is also recorded\* that the first instance of hydraulic packing was due to the flooding of a tributary of the River Dee near Hawarden, Cheshire, which caused sand to be washed into the workings of a colliery which had to be abandoned. On reopening the colliery (The Mary Hey Pit) it was found that the sand had so compacted that the adjacent coal pillars could be safely worked. In 1884 hydraulic stowing was used to extinguish a fire in the Buick Ridge Slope Mine near Shamokin, Pennsylvania. In 1901 the system was introduced into Upper Silesia and soon attracted the attention of German Mining Engineers. The success of the system resulted in its adoption in many countries including Belgium, Austria, Silesia, The Rand, the Pas de Calais, Lenz, Westphalia, Saarbrucken, and India. There were a few places in the United Kingdom where the system was practised with success. Hydraulic stowing is widely practised in Silesia where large quantities of sand

-----

\* Hydraulic Stowing - G. Knox and J.D. Paton,  
Proc. S.W. Inst. Engrs. Vol. XXXVII.

are available in the immediate vicinity of the mines. Between 1900 and 1931 it is estimated that fifty four million tons of sand were used by mines in the Myslowitz district. About one third of the Potash mines in Germany also make use of this system where the method of working is very suitable, as the rooms are 300 ft. long, 45 ft. wide and 30 ft. high.

The practice of hydraulic sand stowing has gradually increased in India from about 1911, when a few mining engineers early realised that large-scale extraction of the valuable seams in the Raneegeunge and Jharia fields would be impossible without the aid of some form of solid goaf packing. The majority of the seams are thick, liable to spontaneous combustion and, in working at depth, to severe and dangerous roof bumps. The question of the necessity of obtaining the maximum percentage of extraction is also vital, as the conservation of these rich seams is of paramount importance to the future of Indian industry. In view of these circumstances and the evidence of wastage of valuable coal in other coalfields, the Government of India introduced legislation constituting a body called the Coal Mines Stowing Board, to control and finance schemes of underground stowing. The Board draws revenue from a royalty on coal sales and at present gives financial aid in obtaining supplies of stowing material to colliery companies who are in a position to benefit most from this help. The magnitude of



these operations and the finance involved is little known outside Indian Mining circles. In the Ranee-gunge field alone the present sand tonnage handled is in the neighbourhood of 2,500,000 tons per year, all taken from a five-mile stretch of the Damoodar river. In the United Kingdom, due to several causes, this system was never popular. Successful operation was, however, carried out in Scotland at Lochhead and Michael Collieries where large tonnages of coal were won in the Dysart Main seam. Hydraulic Stowing was introduced at Lochhead Colliery over 30 years ago and proved economical. Some 350,000 tons of debris were stowed and the system was only abandoned when the flushing pipes reached such a length that it was taking an unduly large amount of water to carry the material. In Coal Mining, hydraulic stowing has many drawbacks amongst which are the effect of water on soft strata, and the added burden of pumping from great depths. Where conditions permit, its use, however, there is still much to commend the system and its popularity is increasing in many parts of the world, more especially those places where ample supplies of sand are available in or near the mining areas.

#### (ii) Modern Plant and Practice

Briefly, the practice of hydraulic stowing can be divided into two main operations:-

1. Surface handling of stowing material to main bunker.
2. Underground conveying of mixtures to the goaf.

In obtaining sand supplies, three main systems are used: manual loading into tubs, Scrapers or Drag Lines and sand pumps. The material is conveyed by aerial ropeway, rope haulage, locomotive haulage, or in favourable cases pumped direct into the pit head bunkers, the capacities of which vary up to 3,000 tons.

As against the recognised standard methods of excavation and transportation on the surface, there is considerable deviation of opinion as to the most suitable method of transportation of sand underground. When small pioneer units were started, tonnages handled were small and rule of thumb methods existed. With the demands for more efficient and economical performances, considerable private research has been carried out by various companies, each setting their own standards. The plant and practice described has been evolved as the result of many years of fundamental research in which the writer and others were engaged and constitutes the latest practice in the hydraulic stowing of sand as proved in the collieries of the largest coal mining company in India.

Where the delivery of stowing material is intermittent, it is essential to have a bunker of large capacity at the point of delivery (generally near the pit top) to ensure continuous stowing, which is essential if cheap casts are to be obtained. The bunkers may be of steel construction or excavated out of the solid ground, the inside being faced with concrete or brick. The sand valves and water pipes are at the

bottom of the bunker and connected to the stowing shaft by an inclined road. A section of a typical bunker is given in Fig. 40.

Where a sand pumping system is used, the sand has to be dewatered before being fed to the main bunker, from which it can be taken to a smaller bunker nearer to the point of stowage. In a recent modern plant for taking supplies of sand from under water the following procedure was adopted. The sand pump consisted of a cast iron casing and cast iron impeller driven by a 125 H.P. low tension motor - the pump, motor and starters being mounted inside a pontoon, supplied by a trailing cable on floats. A 6 H.P. centrifugal pump delivers sealing water to the sand pump suction gland. The suction pipe is submerged and is attached by a special flange to the pump and is raised or lowered by a hand winch. From the delivery side of the pump the sand/water mixture is conveyed by an armoured flexible hose pipe 8" diameter to a line of 8" diameter black steel pipes, also carried on floats. The pipe line delivers sand/water mixture into a conical separator at the rate of 8 tons per minute. This mixture is fed on to screen plates with  $1\frac{1}{2}$ " holes, thus reducing the velocity so that the sand is deposited at the bottom of the separator. This sand is then fed on to a drying belt where the remaining water is squeezed out by rollers, as shown in Fig. 41. The dried sand is then conveyed to a 1,000 ton bunker by a 26" troughed belt conveyor fitted with a tripping gear

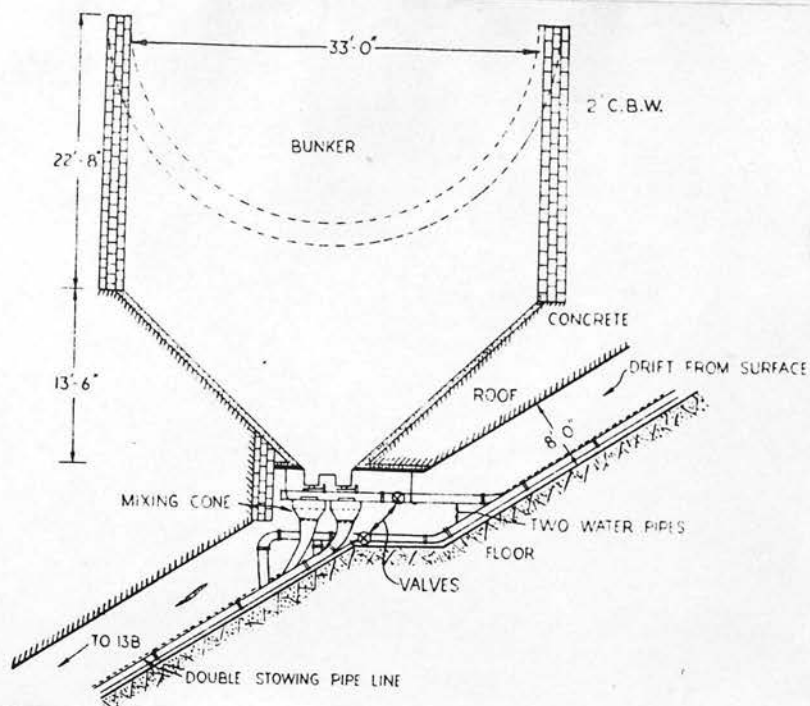


FIG. 40.

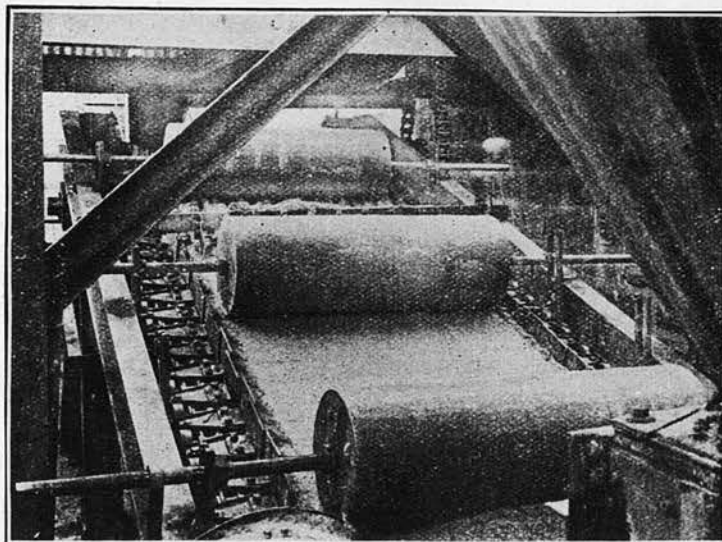


FIG. 41.

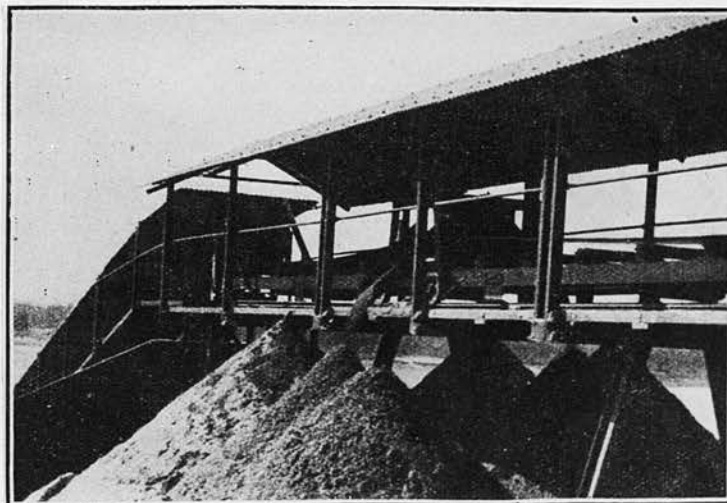


FIG. 42.



as shown in Fig. 42.

From the sand drying plant bunker the sand is automatically loaded into buckets of the aerial ropeway which delivers the sand direct to the pit top bunker. The pit top bunker in this case is of fabricated steel, with a capacity of 250 tons. Underneath this bunker, as illustrated in Fig. 43., are two control chutes which permit the sand to be fed into two mixing cones which feed 5" diameter pipes connected with the stowing shaft ranges by a drift dipping 1 in 3. Water storage tanks are also necessary for recirculation and storage of stowing water, and are usually of a capacity of approx. 150,000 gallons.

With such a layout up to 50,000 tons per month can be handled, with a maximum stowing performance of 200 tons per hour in one pipe.

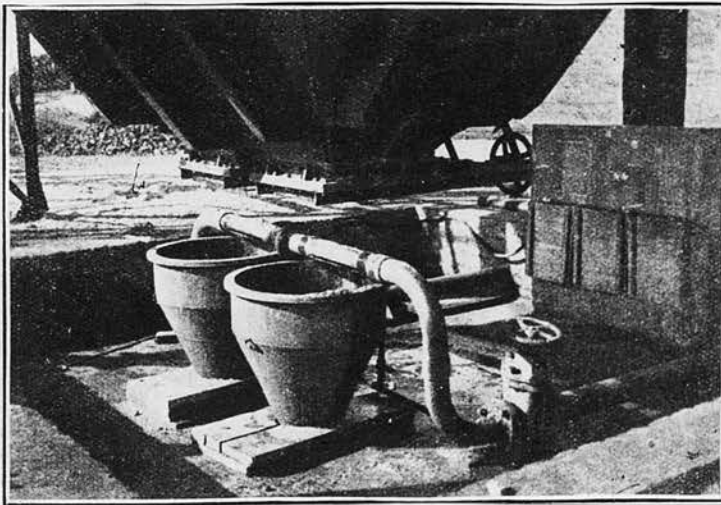


Fig. 43.

(b) PNEUMATIC STOWING

(i) History and Development

The first practical pneumatic stowing machine for use in mines was made in Germany by Torkret in 1927 and consisted of an air tight chamber into which the stowing material was charged and by means of rotating pocketed wheel was fed out into the pipe line, the compressed air blowing the material through the paddles and along the pipes. The working of this prototype was intermittent as the blowing operations had to be stopped whilst the chamber was recharged. This machine was later successfully developed into single chamber and double chamber types. Between 1927 and 1930 several other machines were designed notably the Ollrogge, Miag and Palissa in Germany, whilst an adaptation of the Palissa was developed in this country. This worked on the cannon principle and had one or two short barrels directed towards the goaf, the material being fed into a hopper above the barrels. Air jets at the rear of the barrels blew the material into the goaf. Whilst this type of machine was moderately successful, it was very heavy in air consumption and has since been abandoned.

The Beien machine, first made in 1929, was arranged to give continuous operation. The principle feature was that the material was introduced into the air stream by means of a long horizontal cylindrical valve. In 1935 Herr Brieden and Herr Römer, the

chief designers left Beien to form a new company to manufacture an improved design which consisted of a conical valve to take up wear. The two firms combined in 1944 and the latest Beien-Brieden machines incorporate the improvements made by both firms.

In recent years Messrs. Markham & Co. have manufactured a Blast stower which is based on the Beien principle as is the Powell Duffryn machine. With the Torkret, Beien and Brieden systems the material is fed into the machine which is installed some distance from the face, the material being conveyed by pipes along the roadway, round a bend and down the face to the point of stowing.

(ii) Modern Plant

In the development of these machines, two main types have emerged as successful systems, one employing a rotating paddle which acts as an air valve and feed wheel, fixed horizontally, the other as an air lock chamber with automatically operated doors and combining a feed wheel on a vertical axis. The present machines in use and the capacity are listed below:-

Torkret Single Chamber (German)	120	cu.yds.	per	hour.
Torkret Double Chamber (German)	90	"	"	"
(Beien ) (German)	40	"	"	"
(Brieden)	90	"	"	"
Powell-Duffryn (British)	50-60	"	"	"
Markham Blaststower (British)	90-120	"	"	"

Torkret Single Chamber Machine

This machine is of considerable bulk and may be considered as a static machine suitable for long

term operation from one site, probably serving several districts. Figures 44 and 45 show views of this machine. The air tight chamber is built up of a sufficient number of sections (A) to hold sufficient material (C) for stowing a length of face about 20 ft. The pocket wheel (D) rotates on a vertical axis and is bolted to the underpart of Sections A. A 5 H.P. compressed air engine (E) drives the pocket wheel which distributes the stowing material fed from chamber A, into the stowing pipe (F). The air blast enters the chamber at (G). The chamber A is made airtight by the swing door (H) which is operated by compressed air. Stowing material from the upper bunker (B) is fed by means of a hand operated shutter into chamber A. Charging the machine generally takes place when a face pipe is being removed. The rated capacity is given as 120 cu. yds. per hour. For each cubic yard of pack 100 cubic yards of free air per minute are required at a pressure of 1 to 2 atmospheres. Owing to its bulk, this machine has not found favour in British Mining practice, although a number of machines are in use on the Continent.

#### Torkret Double Chamber Automatic Machine

This machine, designed for continuous operation, is much lower in height than the single chamber type, but has a lower capacity, being rated at 90 cubic yards per hour. The chamber is divided into two air tight sections with communicating doors which



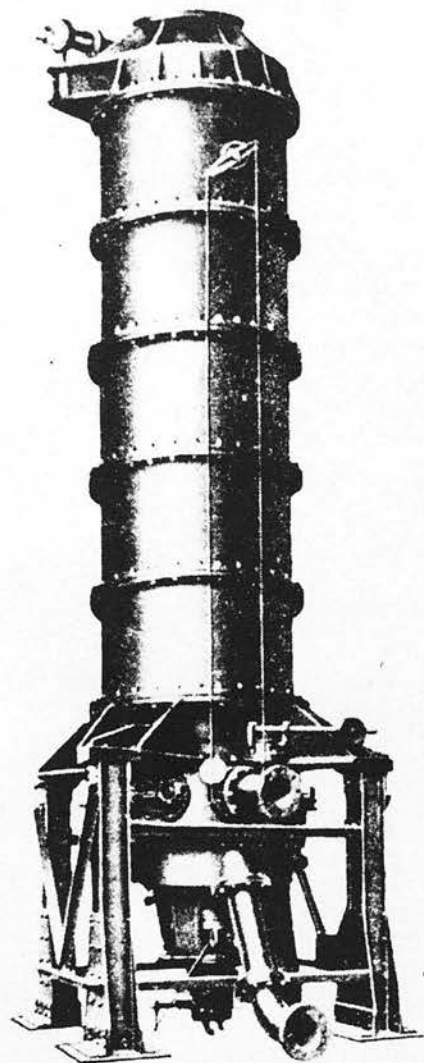
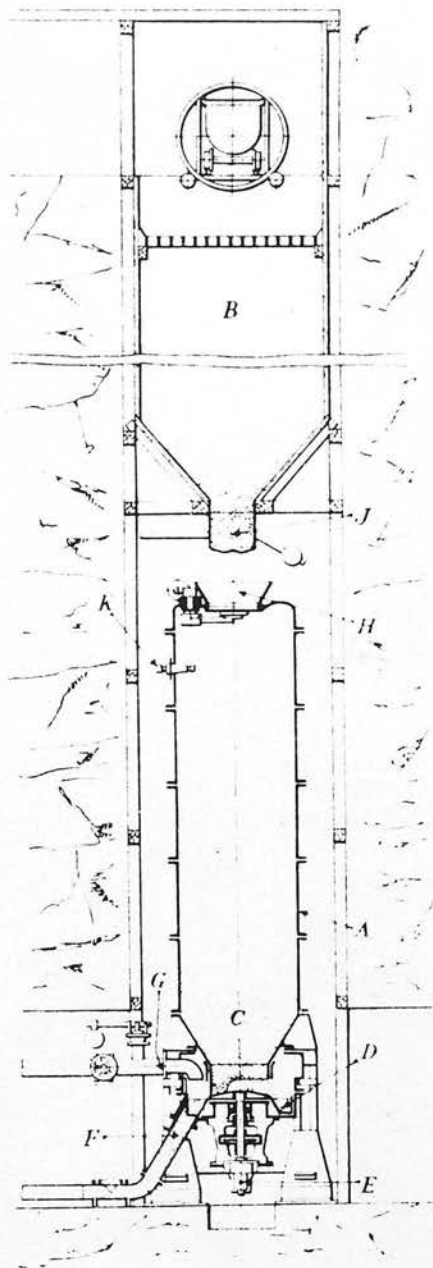


Fig. 44 Torkret machine in section

Fig. 45. Exterior view of Torkret single-chamber machine

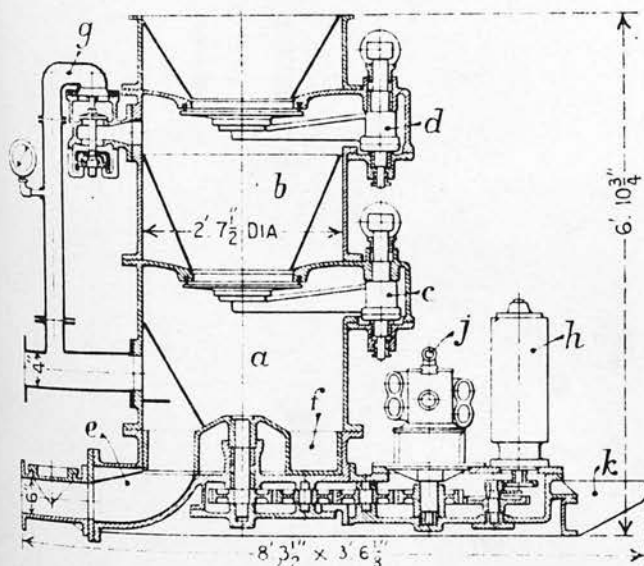


Fig. 47. Section of "Automat" machine

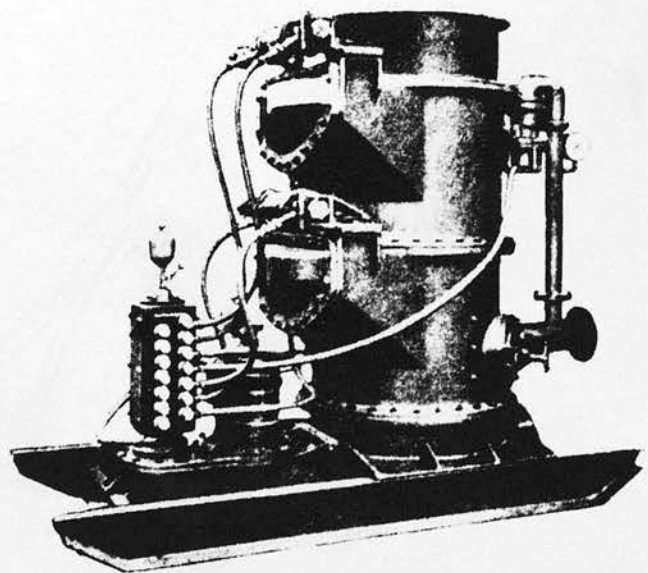


Fig. 46. "Automat" double chamber machine

are operated automatically. Fig. 46 shows an outside view of the machine and Fig. 47 a section. The chambers (b) and (a) are superimposed, the automatic doors (d) and (c) allowing continuous air blast. The pocket wheel (f) regulates the flow of stowing material into the stowing pipe at (e). Valve (g) controls the air supply. The sequence of operations is as follows:- material is fed into the upper chamber (b) and when full the top air tight door is closed. The door (c) is then opened and the material falls into chamber (a) which contains the rotating valve (f). The bottom door is closed as soon as the material has fallen through, the air blast commences and the valve feeds the material into the pipes. Whilst this charge is being stowed the outer top door is opened and receives a further supply of material. These operations are all carried out automatically by a system of cam-operated pilot valves and pneumatic cylinders, the power unit being a 5 H.P. air engine. The machine is built on a sledge and has a height of 7 ft. 2 ins.

#### Beien-Brieden Machine

These machines are used extensively on the Continent and give satisfactory results with the type of material used, but they are not generally suitable for sticky material. Fig. 48 shows a cross-section of the machine. As will be seen, the machine consists of a pocketed valve rotating in a fixed housing. The valve and casing are conical and can be adjusted for wear.

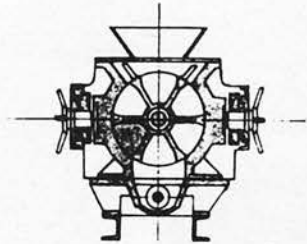
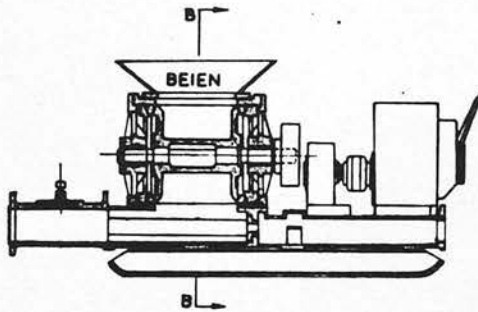
The valve performs two functions, firstly it collects the material from the hopper situated at the top of the machine and delivers it into the air stream and secondly it acts as a valve in preventing the escape of air from the pipe line through the hopper. Directly below the valve the casing is chambered to form part of the compressed air passage and here the material is accelerated by the action of a jet and is carried into the stowing pipes.

#### Powell-Duffryn Machine

The design of this machine is very similar in construction to the original parallel valved Beien. The capacity of the machine has been increased from 30 cubic yards per hour to 60 cubic yards per hour by increasing the size of the orifice. The paddle speed was increased from 20 to 30 R.P.M. It was found that higher speeds gave no advantage, as the pockets did not fill properly and thus reduced the through put. A cross section of this machine (Beien V.H.20) is shown in Fig. 49. Figures taken in South Wales show that an original machine of this type passed 22,000 tons of material before excessive wear and then air leakage caused the machine to be scrapped. At present there are about 40 machines of this type at work in this country mainly in South Wales.

#### Markham Blaststower Machine

The Markham machine is a re-designed Beien



SECTION B B

FIG. 48.—SECTIONAL VIEWS OF MARKHAM AND BEIEN STOWERS.

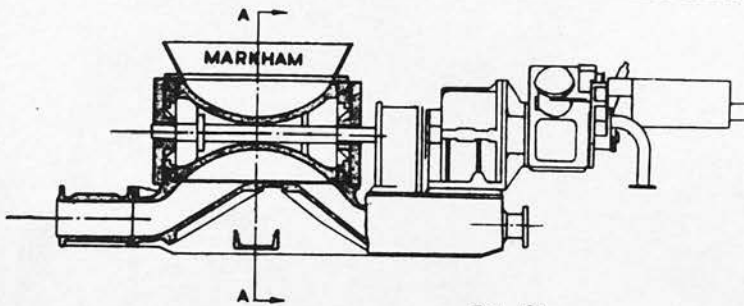
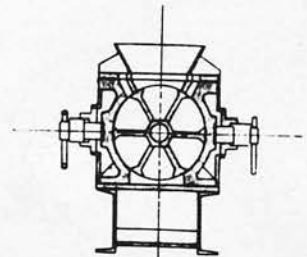


FIG. 51.



SECTION A A

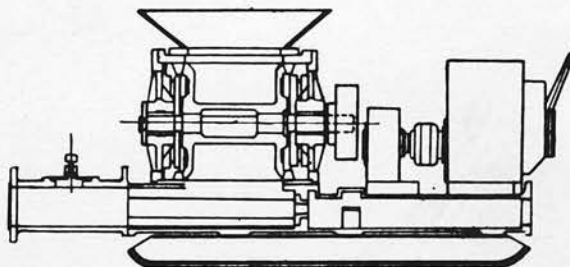
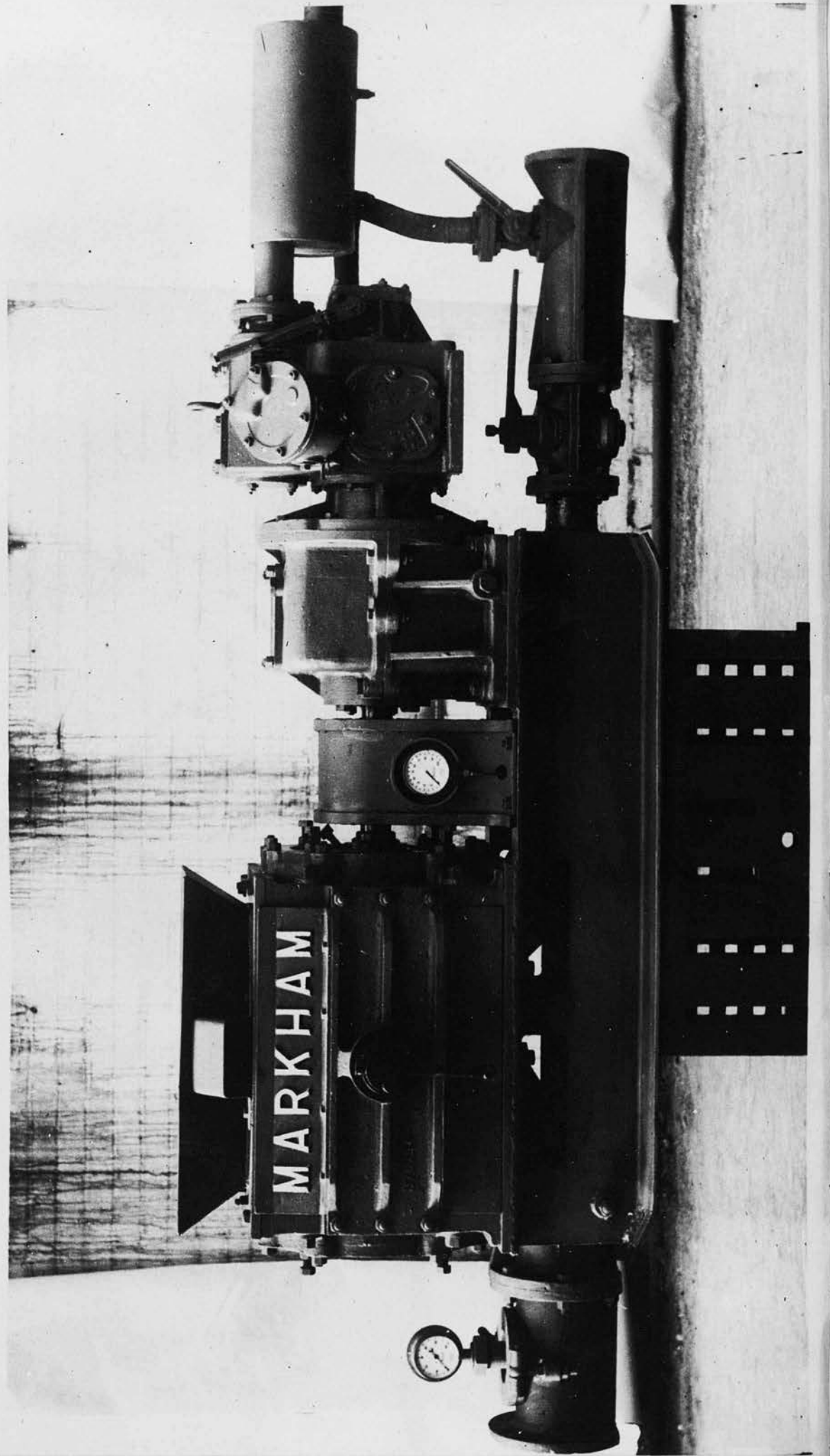


FIG. 49—POWELL DUFFRYN BEIEN STOWER.



FIG. 50.



type machine to enable the successful handling of sticky stowing materials and to increase the through put. A feature of this machine is that the shape of the pockets in the rotating valve has been streamlined and the air directed to sweep up and through the pockets. The casing is provided with a hopper at the top to receive the stowing material, an air conduit to carry the air supply to the stowing pipes and a pocketed rotor valve rotating within the casing. The machine is mounted on fabricated skid frame and the general arrangement is shown in Fig. 50 with sectional view in Fig. 51. The rotor is driven through a gear box by a reversible compressed air engine. A flexible coupling connects the gear box and the rotor whilst the driving shaft has sufficient resilience to absorb shocks. The air supply pipe is carried below the casing and the gear box and a connector is arranged with a control valve to the engine. The main air supply flows through the blast regulating valve to an air jet attached to the underside of the casing and the stream of air is directed through each pocket in turn with the object of keeping them clean. A down-swept pipe attached to the underside of the casing carries the mixture of air and material to the stowage pipes. Details of the Blaststower are as follows:-

Size:- Length 11ft., width 3'2½", height 3'4".

Capacity:- 120 cu. yds. per hour at 40 R.P.M.  
               90                       -do-                       25 R.P.M.

Paddle Wheel:- Length 24", capacity of each pocket 5,167 cu. ins.

Normal revolutions of paddle wheel :- 25

Normal air consumption:- 2,200 cu. ft. per min.

-do- for engine:- 150-200 cu. ft. per min.

Air Pressure in pipe:- 20-25 lbs per sq. ins.

Air pressure driving engine:- 40-50 lbs. per sq. ins.

Capacity of driving Engine:- 10 H.P.

(iii) Power Consumption in Pneumatic Stowing

One of the main points of criticism of this method of stowing is that it is expensive in compressed air. With the modern type of machine, kept in good adjustment and repair and with intelligent handling by the operator air consumption has been reduced. Using 3" size material in 6" pipes up to a length of 400 yards, the consumption should be approximately as follows:-

35 cu. yds. machine	-	1,600 cu. ft. per min.
55 cu. yds. machine	-	2,000 cu. ft. per min.
90 cu. yds. machine	-	2,500 cu. ft. per min.
120 cu. yds. machine	-	3,000 cu. ft. per min.

This gives a cubic ratio of air to material ranging from 100 to 56.

For each type of material there is a certain minimum air velocity required to convey and discharge it at velocity sufficient to give a good pack. The actual pressure required is found in practice by trial and error and varies considerably according to the material and the pipe length. In theory, pressures varying from 5-25 lbs. per sq. ins. are required for 3" material, dependent upon pipe length, but in practice this pressure ranges from 20 to 40 lbs per sq. ins. for pipes up to 400 yards.

Economy in consumption can be obtained by placing an orifice in the air supply column behind the machine. The determination of the size of the orifice required for changing conditions throughout a stowing shift is a matter for investigation and experiments are now being undertaken by the N.C.B. (S.W. Div.) to develop a variable orifice, adjustable to suit changing conditions. This will be actuated by back pressure on the pipe line or an electrical relay device.

It is important to note that for the most economical results from the point of view of air consumption, the equipment should be run at full capacity with the lowest possible air pressure and the maximum material-air ratio. The adjustment of pressure to keep the consumption at a minimum is a point which needs very careful supervision as operators tend to put the full blast at the mains pressure as in this way the liability for blockages is reduced rather than run at lower and more economical pressures. If the material-air ratio is kept low the power costs are not out of proportion.



(c) MECHANICAL(i) History and Development

The considerable extension of the use of conveyors in mining has facilitated the introduction of mechanical methods of transporting and discharging the stowing material from the conveyor to the goaf and a number of ingenious devices have been tried, particularly in France and Germany. The development of these mechanical devices fall into five main classes:-

Throwing Machines - In general principle these machines are fitted with a wheel which rotates on a vertical shaft, the stowing material being fed into the machine and thrown into the goaf. One such machine was constructed on the principle of a coal cutter and hauled itself forward as the goaf ~~was~~ stowed solid. Machines of this type have gradually assumed large overall dimensions and in one case was driven by a 30 h.p. motor and handling material up to 6" cube. It was claimed that the machine was capable of stowing 30-40 cubic yards per hour if continuously fed.

Rotary Blade Machines - Trials have taken place with rotating blades or paddles fixed on a horizontal <sup>material</sup> table, from the conveyor it is caught by the paddles and thrown into the goaf. These machines have the advantage of being able to handle larger material than other types.

Shovelling Machines - These machines are intermittent in action and imitate ordinary shovelling.

They have usually taken the form of a self-discharging attachment on shaker conveyors. One such machine made by Meco consisting of a frame work carrying a shaker elevator trough, at the end of which is an arrangement operated by levers and springs, the actual shaking movement of the engine being used to discharge material into the goaf. By this device lumps of material have been thrown 12-15 ft. by the "shovel". It has the advantage that no additional driving mechanism is required.

Scrapers - These have been tried in Germany for stowing purposes with varying success. Demag has applied the scraper loader and claimed that the plant could handle and pack large size material. The principle is similar to that for conveying coal along a longwall face. A small main and tail hauler drags the scraper, which is automatically charged. The front end of the scraper acts as a ram and tends to force the material more tightly into the goaf. The material is automatically discharged as the scraper is withdrawn. This method of stowage had certain disadvantages such as incomplete filling between adjacent stowing lifts, and there was a cavity of varying depth between the top of the stowage and the roof through which the main rope passed from the scraper to the engine. When the scraper was rammed into the goaf there was always the possibility of sparks being generated and there was constant sparking and heating on top of the stowage by friction of the rope.

High Speed Belts

It was soon realised that hydraulic, pneumatic and "mechanical" methods of stowing had many disadvantages and were generally limited in application, especially the hydraulic method. Some of the early experiments on the use of high speed belts for debris disposal were made in Germany about 1928. Among the first practical machines used in the pit were fast running rubber belts fitted with moulded rubber or angle iron flights to catch the material and enable it to grip the belt. These machines were made by Monninghoff in 1931 and later by Frohlich and Klupfel.

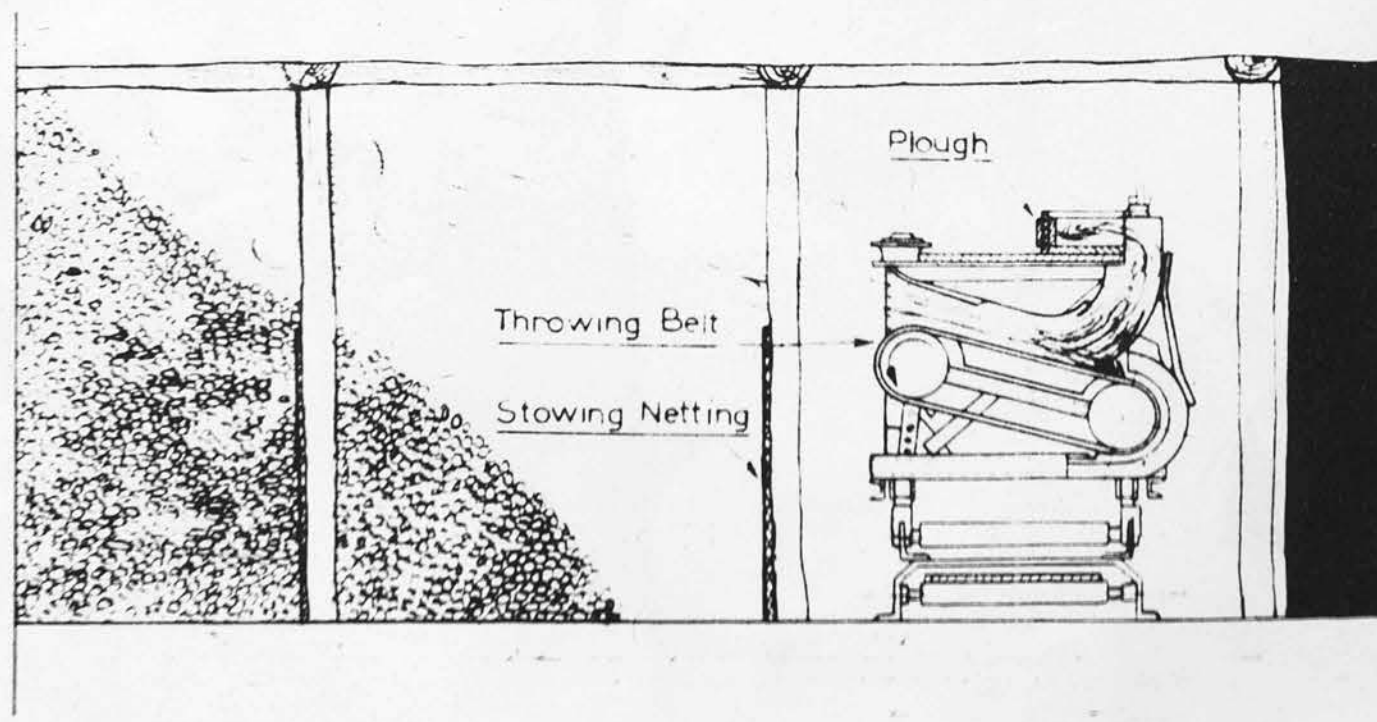
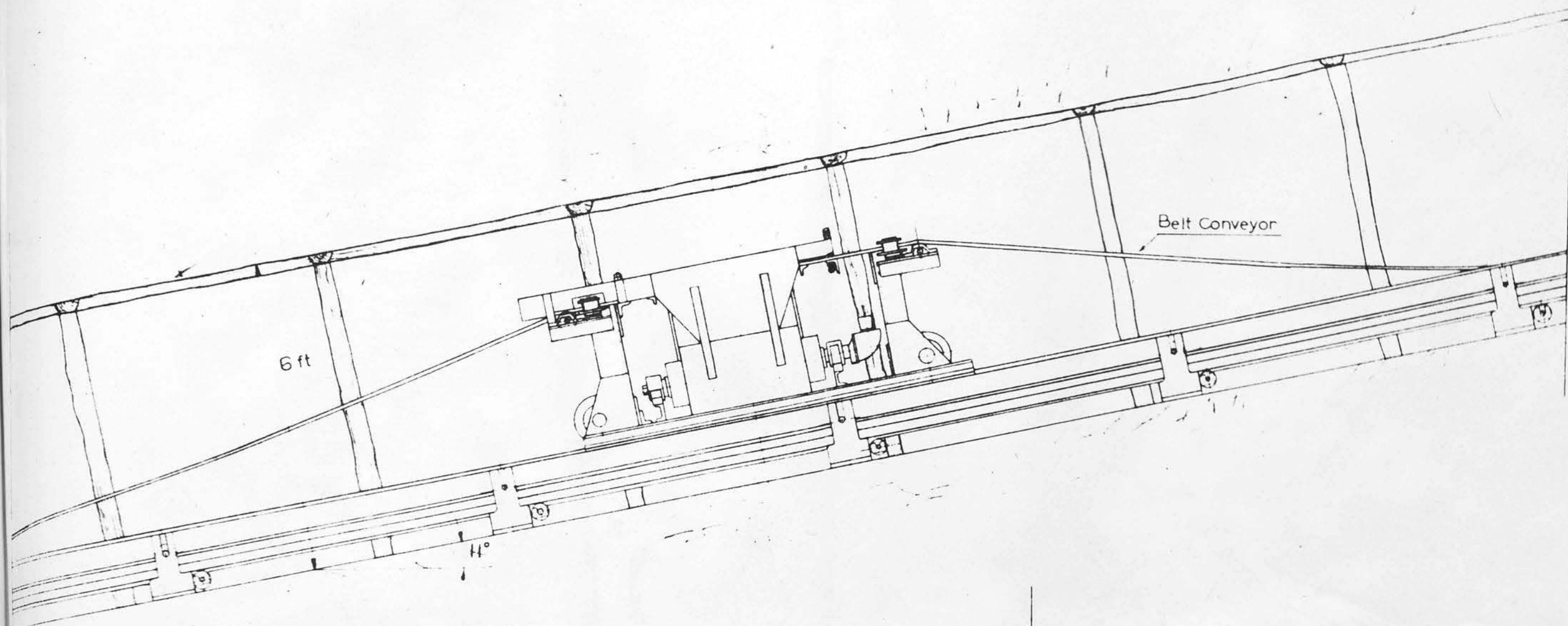
Mechanical stowing in any form is not flexible, the main disadvantages being the handling of the machine in the confined spaces of a coal face and that the material has to be transported and fed into the machine at the stowing site. However, the importance of two factors, viz. the ability to handle a greater variety of stowing material both as regards size and stickiness and a considerably less consumption of power, made the successful application of this method of stowing desirable. In the early models the throwing belt was built on a travelling carriage and the stowing material was ploughed off the face belt into a hopper and thence onto the throwing belt in the direction of the throw. These flat belts were not satisfactory as the stowing material, dropped with practically zero velocity, depended for acceleration

on its weight and coefficient of friction between the material and the belt, thus the material slipped on the belt and was not ejected with sufficient velocity to form a good pack. Fig. 52 shows the arrangement with a flat belt and Fig. 53 a similar machine with an endless belt with moulded rubber flights.

The next development was to increase the slinging velocity of these machines and the problem was partially solved in the "Schleuderversatz", also called the "Kreiselchleuder" or centrifugal slinger, invented by Herr Kuhlmann of Pattberg, Rheinpreussen. As shown in Figs. 54 and 55, the belt is deflected by means of discs bearing on the outside edges of the belt. The material is, therefore, subjected to centrifugal force in order to give it a better grip on the belt surface. The angle of circular contact was in the region of  $94^{\circ}$  and the belt speed 36 ft. per second. This type of machine offers the best scope for future development and modern machines are based on this principle.

Another development in high speed belt throwing machines was in the form of an end on arcing stower used in conjunction with a small loader or a retreating line of shaker pans. The stower was mobile and travelled under its own power, a good feature in the design being that the direction of throw was variable in both horizontal and vertical directions usually in the region of  $15^{\circ}$  right or left and  $30^{\circ}$  vertical. The end on





STOWING MACHINE  
HIGH SPEED THROWING BELT  
FIG. 52.

SCALE: 1/25.

FAST THROWING BELT GDB STOWER WITH COVERED  
IN BOTTOM BELT TROUGHED FACE CONVEYOR.

SCALE: 1 IN. = 1 FOOT.

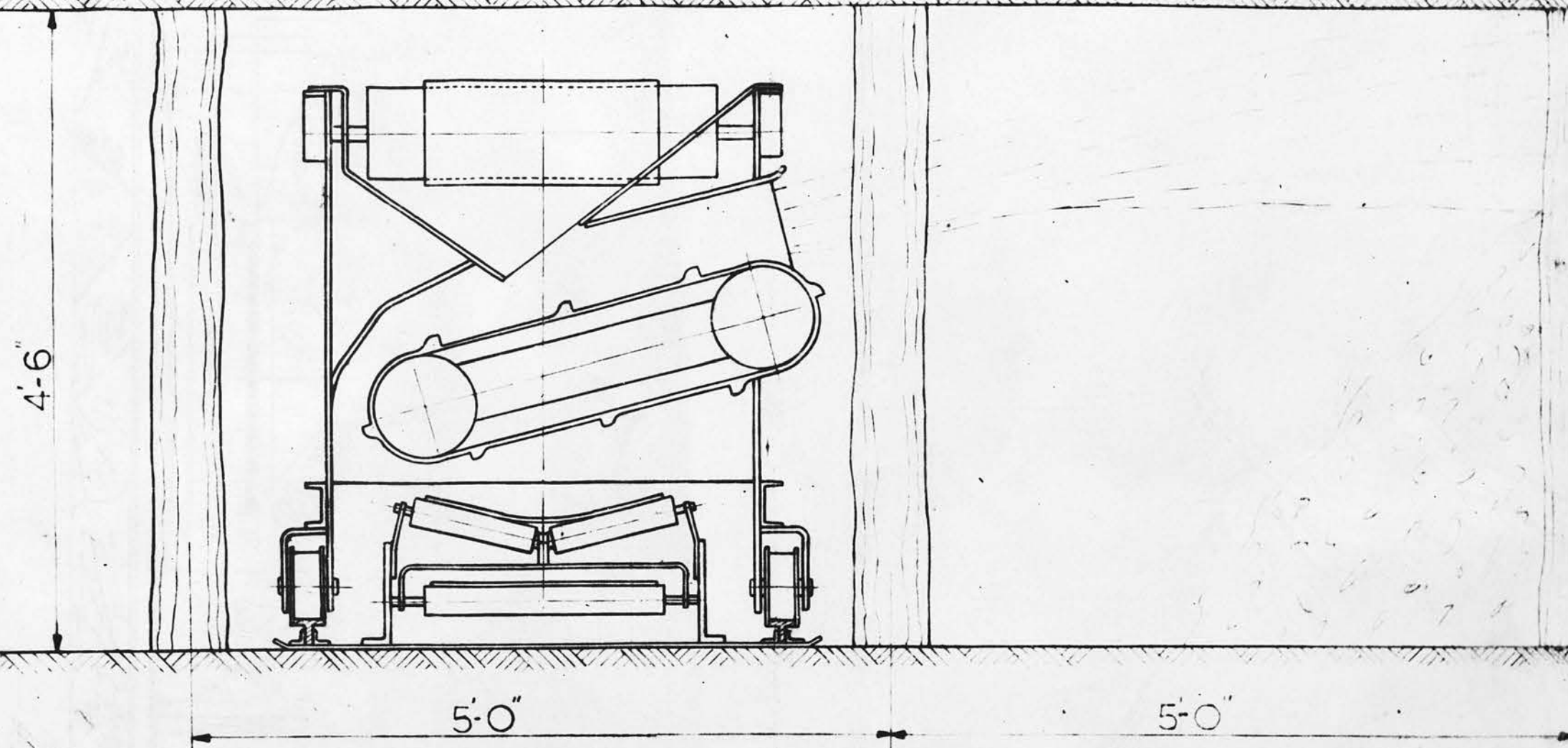
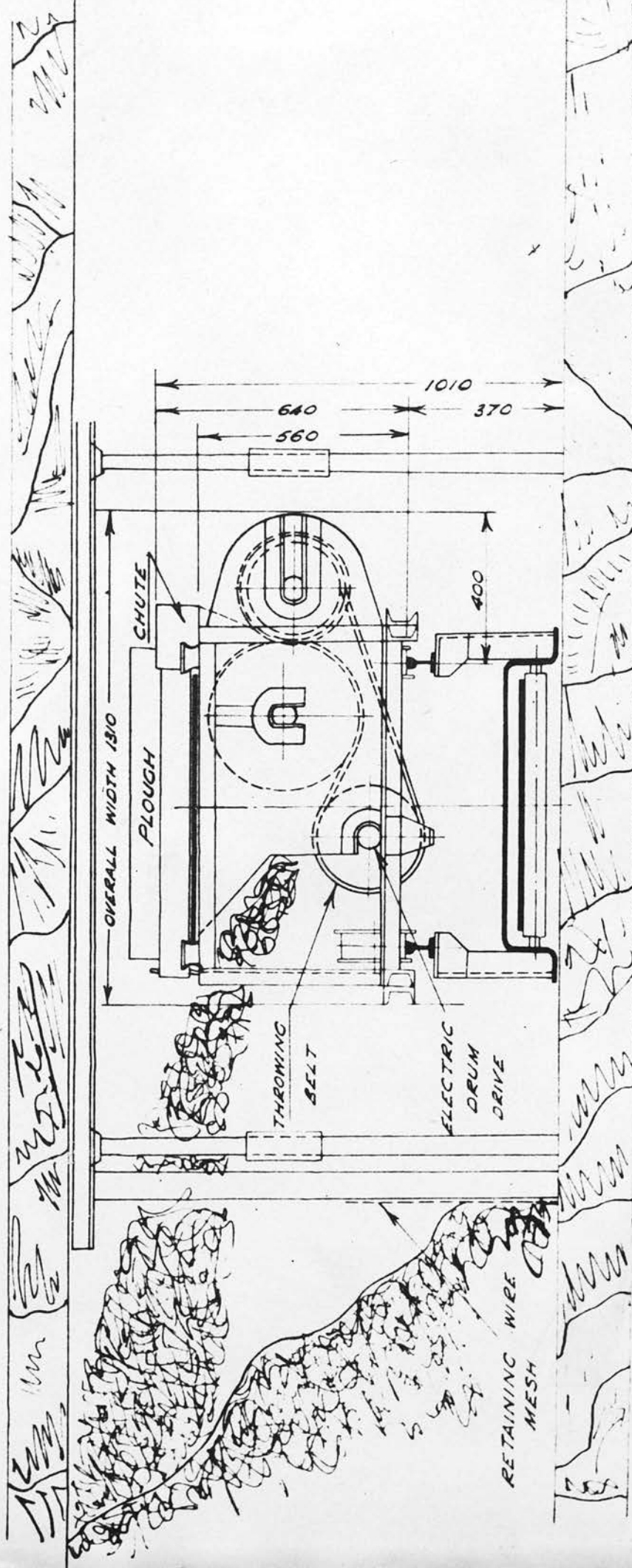


FIG. 53.

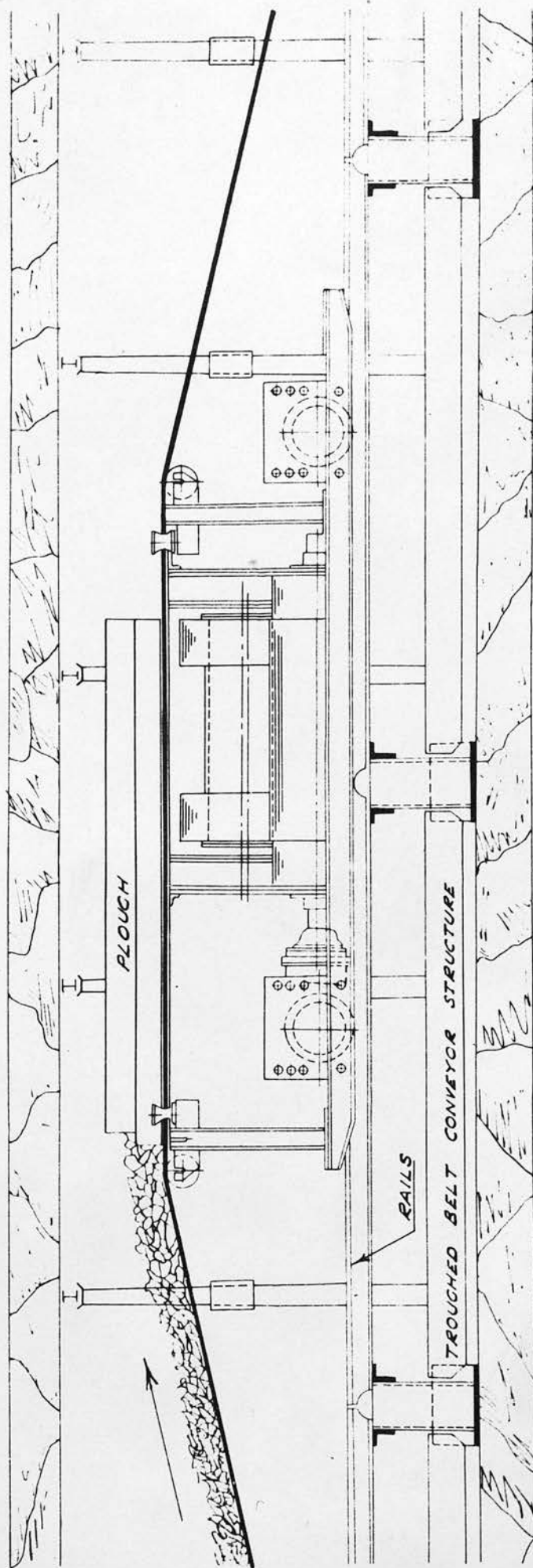


MINE RHEINPREUSSEN

MECHANICAL PACKING MACHINE. DESIGN II.

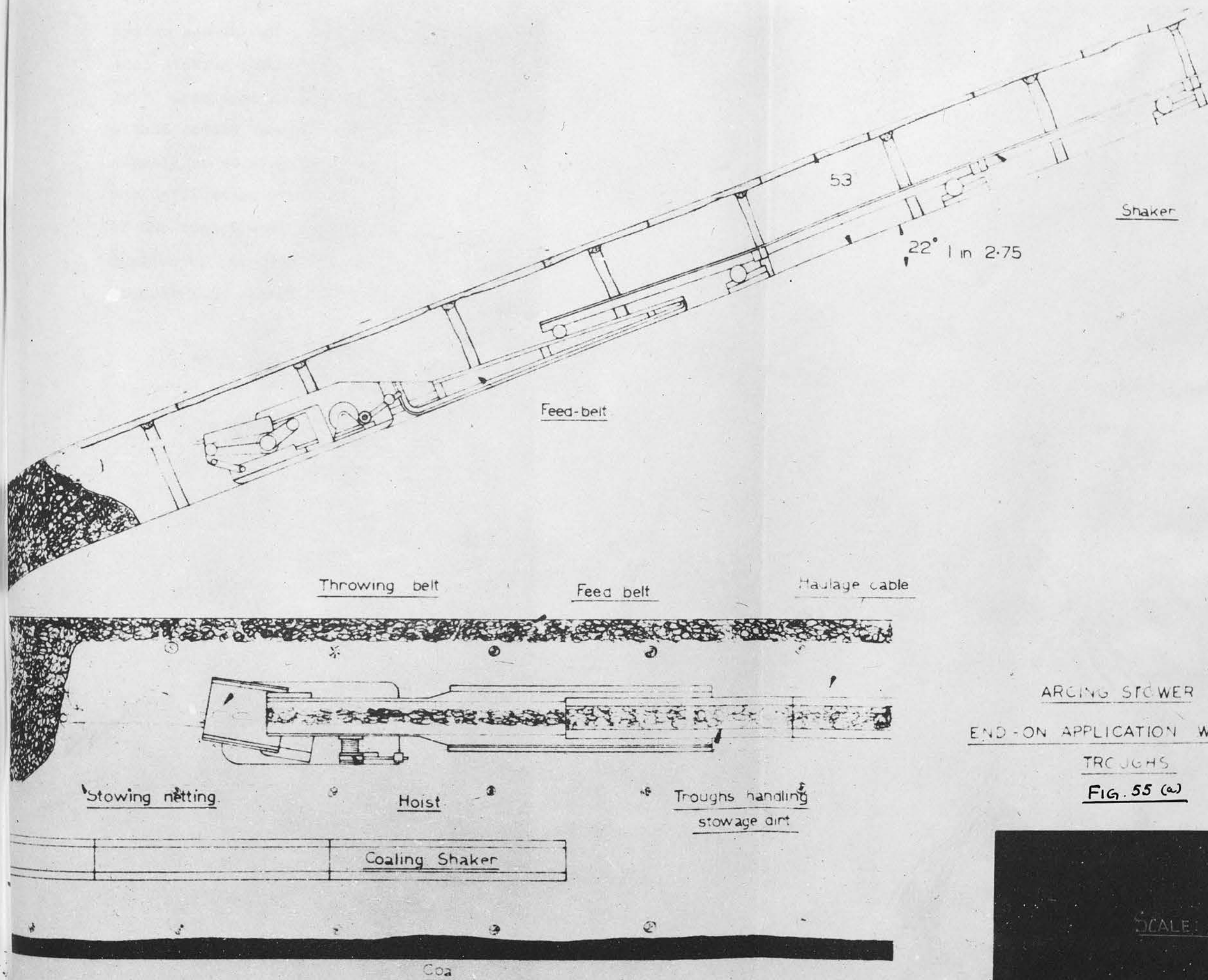
Fig. 54.





MINE RHEINPREUSSEN  
MECHANICAL PACKING MACHINE. DESIGN 11.  
FIG. 55.





ARCING STOWER  
 END-ON APPLICATION WITH SHAKER  
 TROUGHS  
 FIG. 55 (a)

SCALE: 1/50

system has the advantage where the conditions permit, that stowing and coaling can be carried on simultaneously. This type of machine can handle any size of material within reason, without crushing or screening and is capable of stowing up to 200 tons per hour. Whilst the development of this machine has given way in favour of the centrifugal type, there should be a close examination of its possibilities. Fig. 55(a) gives a diagrammatic sketch of the machine and its application.

(ii) Modern Plant

The most recent developments comprise of high speed belts with varying designs of deflecting pulleys, so that material, fed on to the belt and carried to the point where the belt changes direction, is pressed to the belt by centrifugal force, the material thereby gaining a velocity approaching that of the belt before being flung from the machine. Preliminary tests show that material up to 3" can be thrown a distance of 30ft. Some experimental machines have been made which provide an angle of contact of  $150^{\circ}$  as against  $80-90^{\circ}$  of the original German machines, thereby making use of the centrifugal force for a longer period and greatly increasing the velocity of discharge for a given belt speed. To a great extent the problem of imparting a velocity high enough to form a good pack has been solved by the German Schleuder and the present two British machines, Blaw-Knox and Markham. A brief description of these three machines is given.

The Schleuder (Fig. 56.)

This machine is reputed to have a peak capacity of 165 cubic yds. per hour, and although capable of handling this material, the relatively low speed of projection does not form a tight pack. The maximum size of material recommended for the Schleuder is 80 mm. (3.2 ins.), but it will pass larger lumps. As will be seen from the print, the machine consists essentially of a short endless rubber belt, the distance between

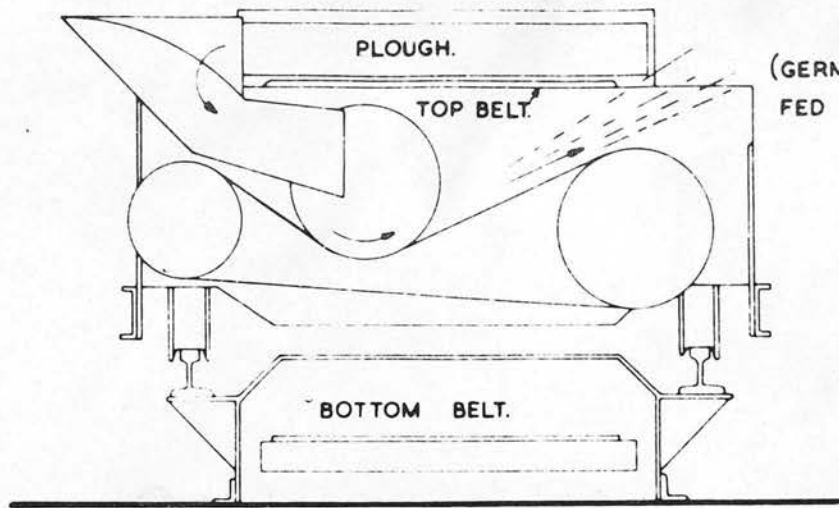


the head and tail drums being about 3'0". Two broad disc pulleys deflect the upper belt downwards about 9". The gap between these two bearing pulleys is approx. 17", the pulleys being 15½" diam. and 5" wide. The stowing belt is 27½" wide and runs at 2,165 ft. per minute (36' per sec.). The driving pulley, which is on the waste side, is 11" diam. and houses the driving motor internally. This motor was designed by Siemens and gives 9 H.P. at 710 R.P.M. The machine is 8' 3½" long and 4' 2" wide and throws a horizontal distance of about 10 ft. Most of these machines were manufactured by Messrs. Frolich and Klupfel of Wappertal. They are not generally recommended for use in this country as they can only be used on relatively flat faces. Where, however, tightness of the pack is not important, this type may find application in debris disposal underground, i.e. where caving is employed and it is desired to gob the dirt cheaply.

#### The Markham Machine (Fig. 57.)

This machine is designed and manufactured by Messrs Markham & Co., Ltd., of Chesterfield and employs the principle of top discharge. The material fed in at the bottom is carried back up and over and finally discharged at the top. The belt lap is about 180° and the belt speed 50 ft. per second. The machine has a high capacity and can handle material as efficiently as pneumatic stowing. It can be adapted for either top or bottom belt feed delivery, the discharge

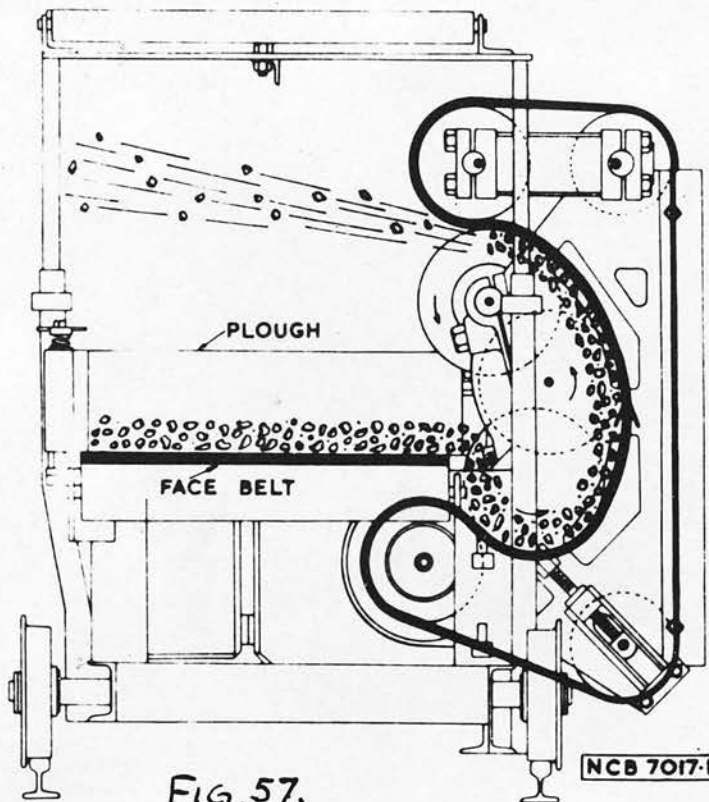




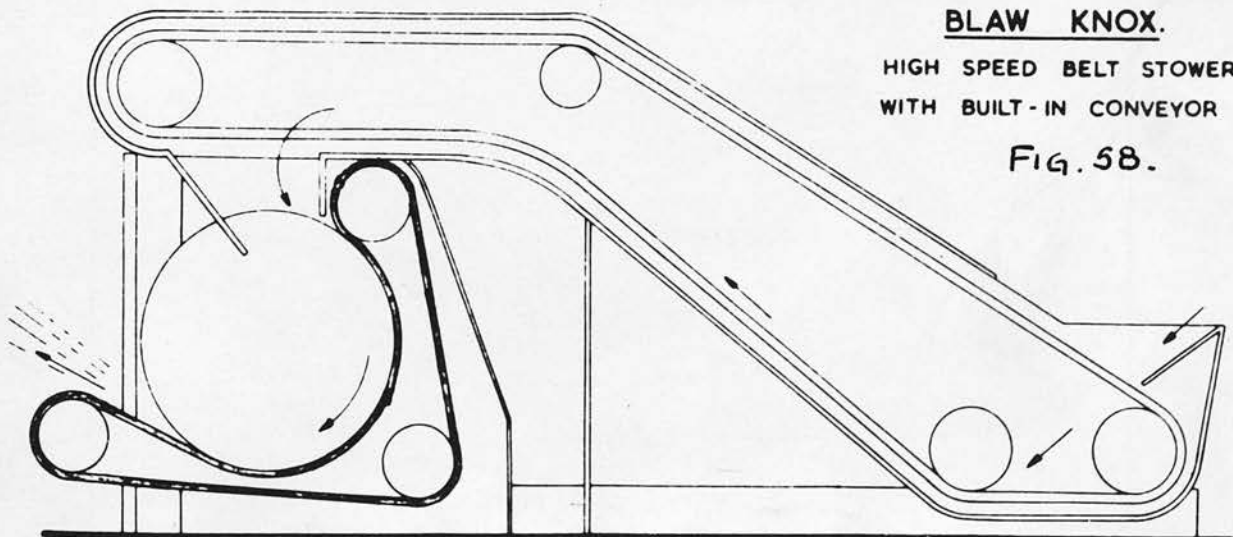
**SCHLEUDER.**

(GERMAN) STOWING MACHINE  
FED FROM FACE CONVEYOR.

**FIG. 56.**



**FIG. 57.**



**BLAW KNOX.**

HIGH SPEED BELT STOWER  
WITH BUILT-IN CONVEYOR FEED.

**FIG. 58.**

belt being 26" wide. The unit is powered by a 10 H.P. A.C. electric motor running at 730 R.P.M. The machine is of compact design being 4' 1" wide with a maximum height of 4'10". The overall length including brake pulley is 9'6". It will be seen from the dimensions that with the present design, application is limited to seams of above 5'0". The machine has been given extensive surface and underground tests in this country and has been designed to traverse on a face belt conveyor.

#### The Blaw-Knox Machine (Fig. 58.)

This machine comprises a high speed belt in the form of a triangle, employing large diameter snub pulleys to depress the belt. The through put and velocity of discharge are high and the stream of material is compact. The machine is shown with a feed belt arrangement, but for underground design this height can be considerably reduced. It would appear that this machine would be more successful used for stowing parallel to the face.

#### (iii) Power Consumption

Mechanical high speed belt stowing systems have a great advantage in power consumption. The present design of belt requires only 10-15 H.P. for stowing and 40 H.P. for supply of material to the machine by conveyor. This compares very favourably with pneumatic stowing which absorbs between 300 and 400 H.P. With

both systems power is only used when actual stowing is in progress.

(iv) Theoretical Considerations in the Design of High Speed Belt Throwing Machines.

High speed have been used in numerous applications and it is only in recent years that attention has been paid to their possible application in underground stowing. The theory of straight belt throwers is relatively simple as compared with belts employing the centrifugal principle. For simplicity and as it is nearer the practical aspect, the behaviour of a particle placed on a belt will be treated as of a non-rolling particle. In practice not only is the shape of a crushed stone irregular, but when crowded on to a belt any tendency to roll is checked by fast moving surrounding material.

Assuming the material to be placed on a straight belt as shown in Fig. 59.

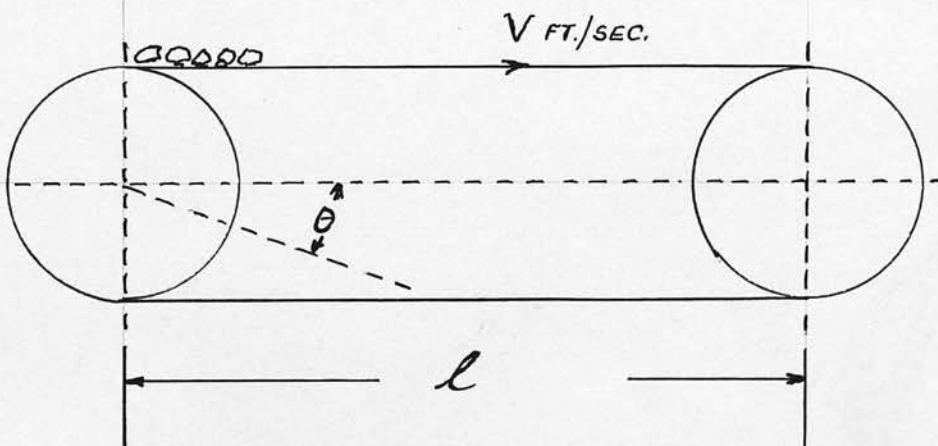
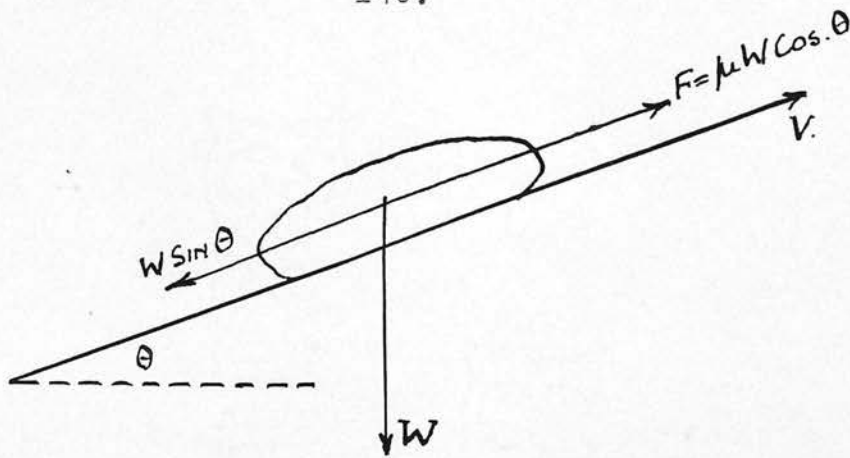


FIG. 59.

The forces acting on any particle are shown in Fig. 60. assuming the belt is elevated through an angle  $\theta$ .

FIG. 60.

The equations of motion of the particle may be written

$$\text{acceleration} = g (\mu \cos \theta - \sin \theta) \text{ and}$$

$$V_1 = \text{particle velocity} = g (\mu \cos \theta - \sin \theta) t$$

$$\text{Linear displacement} = \frac{1}{2} g (\mu \cos \theta - \sin \theta) t^2$$

when the particle velocity  $V_1$  reaches belt velocity  $V$

$$\text{the time taken } t_1 = \frac{V}{g(\mu \cos \theta - \sin \theta)}$$

$$\text{and the distance } S = \frac{V^2}{2g(\mu \cos \theta - \sin \theta)} \dots \dots \dots (1)$$

Theoretically, if varying velocities of projection are not objectionable, any velocity up to  $V$  may be obtained for non-rolling particles if the belt length is sufficient. Equation (1) shows that under ideal conditions this length is

$$l = \frac{V^2}{2g(\mu \cos \theta - \sin \theta)}$$

Taking  $\mu = 0.4$ , with a flat belt  $\theta = 0$  and a belt velocity = 50 ft. per second, this would give a belt length of 388 ft. For underground face work using a transverse conveyor, the length is limited to 3-4 ft.,



so it would appear that the use of a flat belt is not possible without some arrangement for bringing the material up to belt speed within a very short distance. The present trend to get over this problem is to make use of centrifugal type belt throwers, the theory of which is complex and of doubtful practical value because of the variable value of the coefficient of friction between the belt and the material and of the shape, size and type of material to be handled. There is one line of design which may prove to be practical and efficient, using short flat belts, as shown in the diagram below. (Fig. 61.)

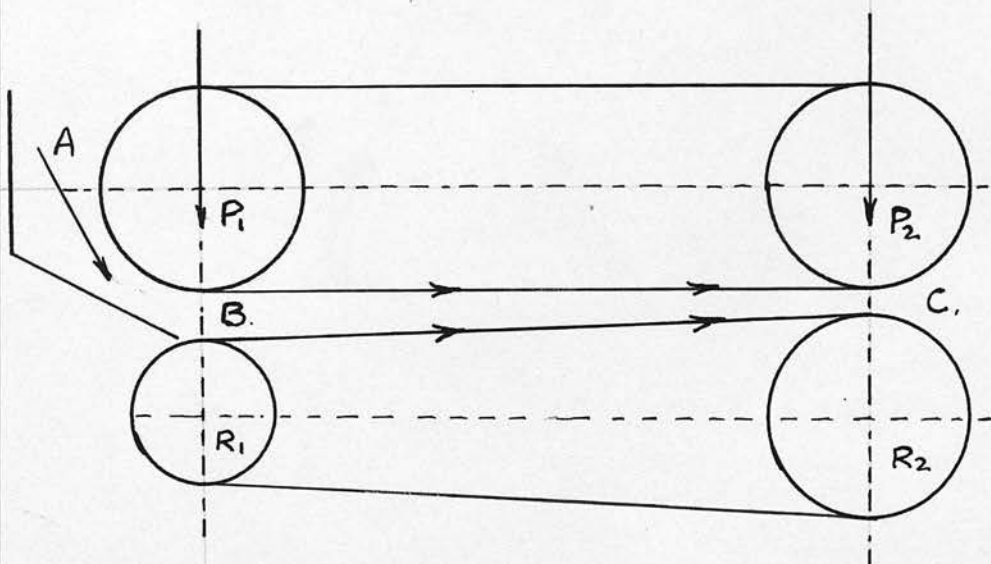


Fig. 61.

Suppose we have two short belts running at the same speed, the bottom belt in a clockwise direction and the top belt in an anti-clockwise direction. The bottom side of the top belt and the top side of the

bottom belt will, therefore, be running in the same direction, at equal speeds, the drive being on pulleys  $P_2$  and  $R_2$ . If the belts converge from B to C and pulleys  $P_1$  and  $P_2$  are spring loaded against the bottom belt then any object entering at A would be gripped at point B and almost instantaneously be taken forward in a mangle action and attain belt speed. Thus in the minimum distance the object would be ejected at C at belt velocity.

This would apply also to a stream of equal sized material, the spring loading ensuring a constant grip throughout the run of the belt. The principle would also apply to close sized material and to a lesser extent to mixed material below a maximum size. The action would then be that material not gripped would be pushed along by the larger sizes and so compacted that it would be ejected at high speed.

The machine could be of compact design, the driving motors being placed in the two driving drums  $R_1$  and  $R_2$ . It is doubtful whether belt wear would exceed the wear already experienced in centrifugal machines as it would be possible to get an equal discharge velocity at a lower belt speed. This method of placing the belts would also lend itself to vertical elevation and also to horizontal arcing a feature not so easily accomplished in other designs.

DESIGN OF INBYE AIR COMPRESSOR(a) COMPRESSOR DESIGN

- (i) General
- (ii) Blowing Pressure
- (iii) Air Consumption
- (iv) General Specification

(b) DESIGN OF A SINGLE STAGE RECIPROCATING AIR COMPRESSOR FOR PNEUMATIC STOWING IN MINES

## (i) Practical Design Data:

- Capacity
- Working Pressure
- Drive
- Cooling
- Dimensions
- Air Temperatures

## (ii) General Considerations

## (iii) Speed

## (iv) Horse Power

## (v) Design of Electric Motor

## (vi) Piston Stroke and Piston Speed

## (vii) Cylinder Diameter and Number of Cylinders

## (viii) Arrangement of Cylinders

## (ix) Crank Arrangement

## (x) Cooling Water

## (xi) Air Reseiver

## (xii) Air Inlet Filter.

(c) GENERAL DESCRIPTION OF PROPOSED COMPRESSOR

(a) COMPRESSOR DESIGNGeneral

It can be assumed that, except in limited circumstances, an air compressor for pneumatic stowing should be an inbye machine. Those circumstances, where a group of collieries are powered by compressed air generated at a central station and used extensively in various phases of mining, are few and, in general, it is necessary for the development of this type of stowing that the question of inbye air compression should be considered. At the present time all inbye compressors are designed to give a comparatively small quantity of air at pressures ranging from 80-100 lbs. per sq.in.

The main advantages of inbye compression are that the high transmission efficiency of electricity is taken advantage of to a greater extent than if air power were generated on the surface. The compressors are always installed close to the place where the air is to be used and hence the transmission losses are small. The main disadvantages are the difficulty of providing an adequate supply of clean cooling water, (although in low compression this is not serious), and the prevention of dust getting into the cylinders with the air. Underground air, especially in deep and dry pits is often extremely dusty and an efficient air filtering arrangement is essential.

There is no reason why in pneumatic stowing the advantage should not be taken to use the principle of



the hot transmission system. In this system the air is compressed in unjacketed cylinders and is taken straight to the machine and is used without cooling. The advantages of such a system, if successfully carried out, are that a larger volume of air is obtained for a given working pressure and the possibility of using the air expansively.

It is not likely that a general application of this system would be desirable, but a compromise might cover all practical conditions. That is, to generate and supply warm air to the stowing machine, the cooling only being carried out to such an extent as to ensure reasonably efficient compression in the compressor.

#### Blowing Pressure

The pressure of compressed air required for pneumatic stowing varies from 1 to 4 atmospheres and depends upon several factors, viz:

1. Type of stowing material used.
2. Rate of stowing.
3. Length of stowing pipe and number of bends.
4. Diameter of pipe.

Tests carried out in various installations give as an average 40 lbs. per sq. in. as a suitable blowing pressure.

#### Air Consumption

The air consumption also varies the factors stated under Blowing Pressure. The original Beien machine (V.H.20) running at 13.5 R.P.M. and having a capacity of 30 cu. yds. per hour required 1,350 cu. ft. of free air per min. at a pressure of 30 lbs. sq. in.

when fitted with a  $1\frac{1}{2}$ " nozzle. The same type of machine with the same size nozzle running at a S. Wales mine at 20 R.P.M. and having a stowing capacity of 42 cu. yds. per hour, required 1,890 cu. ft. free air per min. at a pressure of 30 lbs. per sq. in. A Blaststower machine of 60 cu. yds. capacity would require about 2,500 cu. ft. per min., whereas a machine of 90 cu. yds. per hour would require about 3,000 cu. ft. of free air per min. With the recent improved design of stowing machines as regards air leakage and performance, it is desirable that an inbye compressor should have a capacity of 3,000 cu. ft. per min.

#### General Specification

After a comprehensive survey of past work and possible future applications, the conclusion was reached that a compressor for general use in pneumatic stowing should possess the following broad characteristics:

The equipment should be portable or semi-portable, with cooling arrangements, having a minimum capacity of 3,000 cu. ft. free air per min. at 40-60 lbs. pressure and be so designed for use near the coal face.

The compressor should be compact and be so constructed as to be easily dismantled and erected. It must be so designed to give a steady torque on the motor driving shaft by radial or horizontally-opposed arrangement of the cylinders and give a steady, uniform delivery of air.

In the past no inbye compressor of such a

capacity has been manufactured, due to cooling difficulties and orthodox vertical designs. As the design of such a unit is a prerequisite of successful underground pneumatic stowing, a machine was designed by the writer and Messrs Alley and Maclellen Ltd., of Glasgow to the general specifications set out above embodying all the latest advances in compressor design and ancillary equipment.

(b) DESIGN OF A SINGLE STAGE RECIPROCATING AIR COMPRESSOR FOR PNEUMATIC STOWING IN MINES

(i) Practical Design Data

Capacity - 3,000 cu. ft.

Working Pressure - 40 lbs. per sq. in.

Drive - The compressor to be direct coupled to a flame proof electric motor and all to be mounted on a semi-portable skid base.

Cooling - Jacket cooling water to be cooled by radiator type cooler with fan driven from main motor shaft. An air receiver to be mounted on a separate base and coupled up to the compressor by flexible piping.

Dimensions - The whole assembly to be so constructed that it can be transported and erected in a roadway 12'0" wide and 8'0" high.

Air Temperature - Air delivery temperature to be such that no inconvenience to operators on the face is felt. The temperature underground may be taken to be about 70°F and 40-50% humidity.

(iii) General Considerations

A standard machine of this capacity for use at the surface would be two crank, double acting and would be about 12 ft. high, therefore this type is ruled out on account of the height and the weight and bulk of the individual parts. Since the individual components must be of such size that there would be no difficulty of transporting underground a machine with multiple cylinders is indicated. Consideration was given to a Vee arrangement of cylinders but this was abandoned after trial and finally a machine with horizontal opposed cylinders was adopted as the arrangement best suited



to the available space.

(iii) Speed

A moderate speed is desirable as the foundation would not be permanent, but skid mounted. A speed of 375 R.P.M. was chosen, this being the synchronous speed for a motor with 8 pairs of poles operating on a periodicity of 50 cycles per second.

(iv) Horse Power

The driving motor would be required to have a B.H.P. of 400, as calculated below:

Theoretical H.P. (adiabatic compression)

$$\begin{aligned}
 &= \frac{144 P_1 V_1 N}{33000(N-1)} \left\{ \frac{(P_2)^{\frac{N-1}{N}}}{(P_1)} - 1 \right\} \\
 &= \frac{144 \times 14.7 \times 3000 \times 1.4}{33000 \times 0.4} \left\{ \frac{(54.7)^{\frac{0.4}{1.4}}}{(14.7)} - 1 \right\} \\
 &= 672 \left\{ \frac{3.73^{286}}{3.73} - 1 \right\} \\
 &= 672 \times .46 \\
 &= 309
 \end{aligned}$$

For a machine of this type the B.H.P. required to drive after making allowance for valve losses, mechanical friction, etc., should be approx. adiabatic H.P. plus 30%; therefore

$$\text{Driving H.P.} = 309 \times 1.3 = 401$$

say 400

(v) Design of Electric Motor

(v) Design of Electric Motor

For A.C. supply the driving motor may be of several types; squirrel cage, slip ring induction, auto synchronous or variable speed. For a compressor of this size it was considered advisable to fit a totally enclosed, frame ventilated, flame proof, wound rotor, induction type motor. Due to the size and weight of the motor the following points were considered for a reduction in size:-

1. Higher speed. The objection to this is that more cylinders would be required since the stroke would have to be reduced and the stroke diameter ratio would become inconvenient if the cylinder diameters are retained. The other alternative would be a high speed motor with a reduction gear - which would increase the bed plate size.

2. To fit motors of 200 H.P. at each end of the compressor. With the present design there is no serious obstacle in this except that the question of torsional vibrations would have to be thoroughly investigated. It was considered that the 400 H.P. motor could be handled even if necessary by transporting rotor and stator separately.

(vi) Piston Stroke and Piston Speed

The speed and power having been decided, it is necessary that the piston stroke must be fixed so that the over all length of the machine over the cylinders will be suitable for the space available and at the same time the piston speed must be reasonable. From experience it was decided to limit the piston speed to 700 ft. per min. which was considered a very moderate figure.

Therefore, since piston speed

$$= 2 \text{ R.P.M.} \times \text{stroke in feet}$$

then stroke in inches

$$= \frac{700 \times 12}{750}$$

$$= 11.2 \text{ inches}$$

By making the stroke 11" the piston speed is found to be 688 ft. per minute and checking the length of the machine over the cylinders after making a skeleton arrangement the dimensions are within the specification.

(vii) Cylinder Diameter

The capacity per cylinder will determine the number of cylinders required. The capacity is limited by the loads that can be carried on the crank pin and the piston pin and by the weight of the piston, since inertia loads have also to be taken by the bearing areas.

A reasonable ratio of diameter and stroke would indicate a cylinder of about 16" diameter.

Assume a volumetric efficiency of 80%. This

figure will be checked later when details of clearance space are known and, if necessary, adjustments will be made.

Capacity per cylinder in cu. ft.

$$= \frac{(.7854d^2 \times S \times \text{R.P.M.} \times \text{Effic.})}{1728}$$

where d = diam. of cylinder in inches,  
S = stroke in inches

$$= \frac{.7854 \times 16^2 \times 11 \times 375 \times 80}{1728 \times 100}$$

$$= \underline{384 \text{ cu. ft.}}$$

Number of cylinders required

$$= \frac{3000}{384} = 7.8$$

$$= \underline{8 \text{ cylinders required}}$$

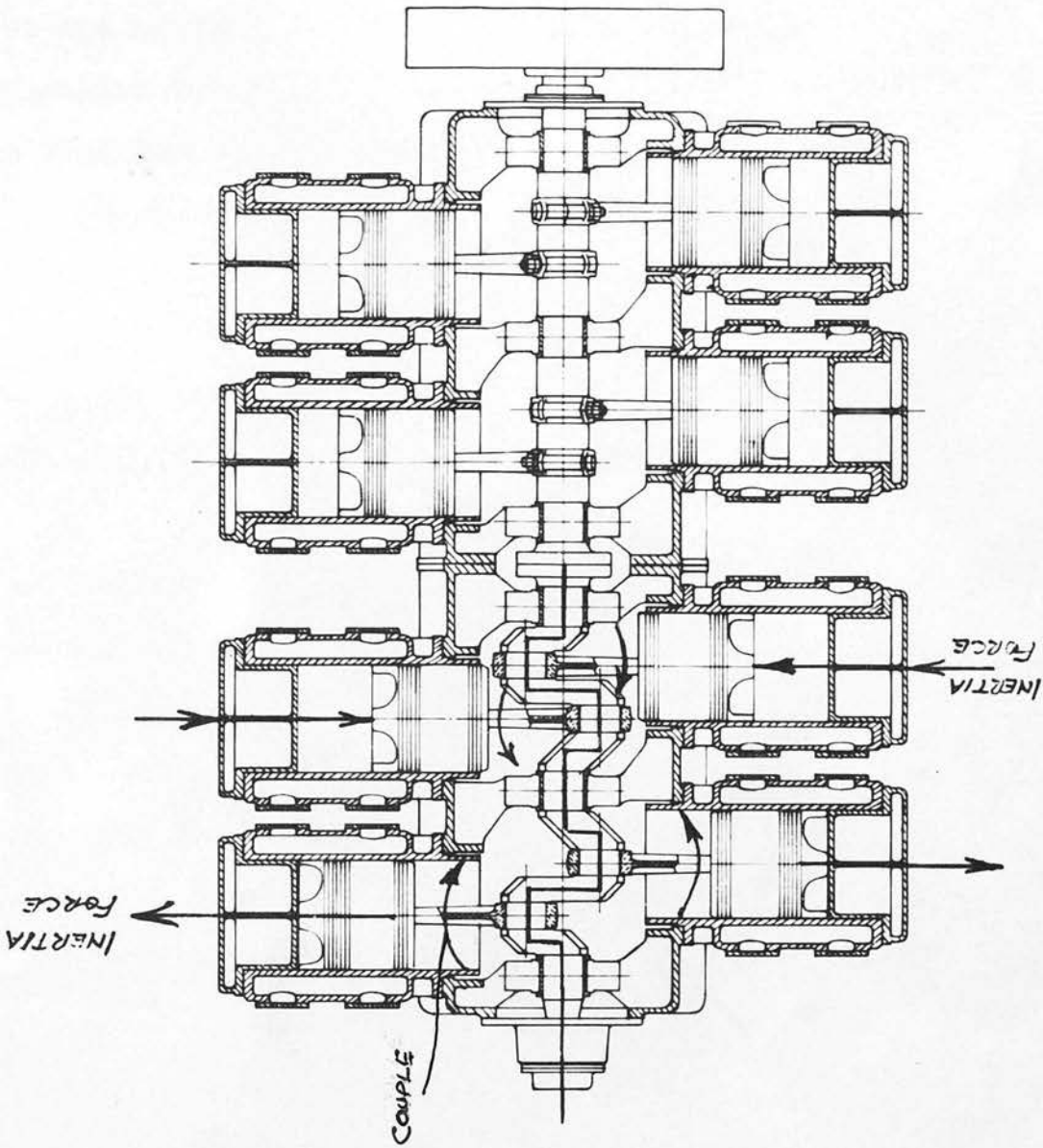
#### (viii) Arrangement of Cylinders

Since the overall dimension of individual parts or assemblies must be kept as small as possible, it was decided that the compressor should be built in two similar sections, the crank cases being bolted together by an oil tight joint and the shafts by a spigotted and bolted flange coupling. Each section would, therefore, have four cylinders in two opposing pairs.

#### (ix) Crank Arrangement

Since the machine is intended to be run on a skid base not necessarily bolted down on a deep foundation as most stationary reciprocating machines are, the fact that we have two sets of four cranks enables us to arrange that the inertia forces oppose and balance





PROPOSED AIR COMPRESSOR FOR PNEUMATIC STORAGE.

CAPACITY — 3000 CU. FT. FREE AIR PER MINUTE.  
 40 LBS PER SQ. IN GAUGE PRESSURE.  
 370 R.P.M. 400 B.H.P.

SCALE 1" = 1'-0"

ALLEY & MACLELLAN LTD  
 POLMADIE, GLASGOW.

FIG. 62.

DRAWING NO. T 9674.

out one another and couples can also be balanced out by making couples act in opposite directions, as shown in Fig. 62.

By arranging the cranks in the way suggested, the fluctuation of energy will be reduced and consequently a comparatively light flywheel will be sufficient to give smooth running with a minimum of cyclic irregularity. Since the inertia forces and couples are balanced, the machine will run without vibration and with the minimum of foundation and holding-down arrangements.

The complete compressor can now be schemed out in detail somewhat as shown on Drawing No. T.9674 keeping in mind that such details as the air valve assemblies, oil pump, air governor and control gear, etc., being standard for other designs, are capable of being used in this application.

(x) Cooling Water

The next thing to be considered is the provision of circulating water for the cylinder jackets and the arrangements necessary to dissipate the heat gained by the water in keeping the air cylinder cool.

From experience it is considered that it would be necessary to deal with 625 B.T.U. per H.P. per hour and therefore in this case, with a total of 250,000 B.T.U. per hour, and radiators capable of dealing with this total rate of dissipation are shown on the proposed compressor Drawing No. T. 9673. Two fan cooled radiators driven from the water circulating pumps are fitted compactly at the motor end of the compressor. These radiator coolers are usually only fitted to small type compressors, but even under pit conditions this type will efficiently deal with the temperature ranges likely to be found. Their construction is convenient where where compressor drive and cooler are required to be fitted in a confined space.

The Drawing also shows each bank of cylinders supplied by a small motor driven water pump of 835 gallons per hour capacity situated at right angles to the radiator type water coolers, the fan drive being shown from the motor coupling. Water circulated in each cylinder bank at the rate of 835 galls. per hour will limit the rise in water temperature to 15°F. and the radiator will have cooling surface and capacity of fan capable of cooling this quantity of water from 100° to 85°F. with air at 70°F. viz. 250,000 B.T.U. per hour.



-TABLE 20-  
PNEUMATIC STOWAGE

MICHAEL COLLIERY

	Monday			Tuesday			Wednesday			Thursday			Friday		
Total Hours Compressor running.	6½ hrs.			7 hrs.			7 hrs.			7 hrs.			7 hrs.		
(Temperatures are in degrees F.)	Immed. after Starting.	Mid Shift	Immed. before Stopping	Immed. after Starting	Mid Shift	Immed. before Stopping	Immed. after Starting	Mid Shift	Immed. before Stopping	Immed. after Starting	Mid Shift	Immed. before Stopping	Immed. after Starting	Mid Shift	Immed. before Stopping
Temp of water in Tank	86	96	102	92	106	111	96	107	109	86	103	109	99	107	110
Temp of water entering Comp.	85	93	97	91	105	110	96	107	109	87	100	109	99	106	110
Temp of water Intercooler	84	100	101	90	129	131	81	125	128	81	118	129	82	123	131
Temp of water leaving Comp.	84	101	106	90	135	139	82	136	138	83	130	140	83	134	140
Temp of High Press. Cylinder.	78	168	238	82	234	238	88	242	240	88	220	234	87	234	230
Temp on Air Pipes leaving Comp.	83	105	171	80	179	180	81	185	185	82	172	185	82	183	178
Temp of Air in Compressor House.	77	90	97	80	101	102	81	100	102	82	100	103	82	99	101
Temp on Air Receiver.	84	96	133	84	166	171	83	173	175	84	164	176	84	171	171
Temp on Pipes Top of Hopper.	84	82	81	74	72	85	69	80	77	69	81	76	69	78	76
Temp on Pipes at Face bend.	78	74	74	76	76	72	73	72	72	72	71	70	73	79	74
Temp of Face Air current when actually stowing.	71	71	71	71	71	71	70	72	71	71	72	72	71	71	71
Quantity of Air passing along Face when actually stowing	4800 c.ft.			4800 c.ft.			5160 c.ft.			5160 c.ft.			5160 c.ft.		
Depth of water in tank.	2' 8"	2' 7"	2' 6½"	2' 6"	2' 5¼"	2' 4½"	2' 3¾"	2' 3"	2' 3½"	3' 4½" (water added)	3' 4"	3' 2½"	3' 2¼"	3' 1¼"	3' 0¼"
G.P.M. in Circulation															
Temp of 'make-up water after running say 5 minutes.															

NOTE: Temp. of Air on Face 15 mins. after compressor stopped was 71° and the Quantity was 4800 cub. ft. per min.

Quantity of water required to reduce temperature  $15^{\circ}$

$$= \frac{250,000}{15 \times 10} = \text{approx. } 1670 \text{ galls.}$$

$$= \underline{835 \text{ per pump per hour}}$$

For interest, a table showing underground temperatures around the inbye air compressor plant at Michael Colliery is given. The compressor is a standard machine not designed for inbye work and is an Ingersol Rand Type EXVH 27 having a rated capacity of 1900 cu. ft. per min. to 75 lbs. per sq. in. and is driven by a 305 H.P. motor. (TABLE 20)

(xi) Air Receiver

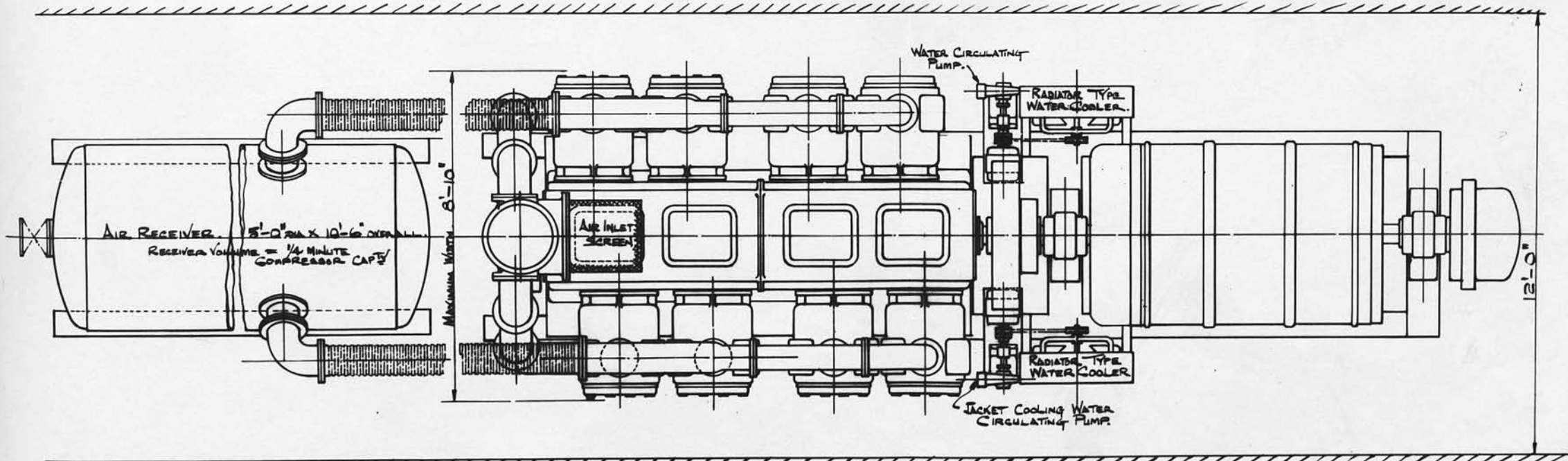
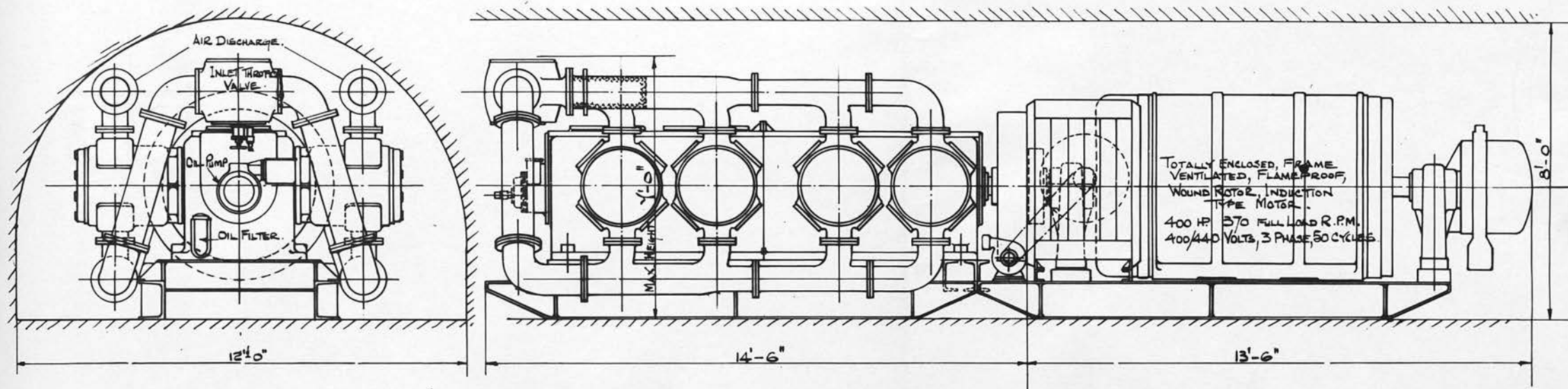
The receiver is inserted to damp out pulsations and since the machine is a multi-cylinder arrangement with half the cranks at  $90^{\circ}$  to the other half, the discharge pulsations will not be very serious in comparison with, say, two crank, double acting compressor. The receiver shown on Drawing No. T. 9673 is capable of storing  $\frac{1}{4}$  minute capacity of the compressor and this should be sufficient to prevent variations in pressure due to pulsations and also to give that volume of storage necessary to enable the pressure governing gear to function satisfactorily. The receiver is 5' 0" diam. and 10' 6" long fed by two inlet pipes of 8" diam. and constructed according to British Standard Specification for riveted receivers. The fittings include pressure gauge, safety valve and drain cock. The receiver is mounted on a separate skid plate base and connected to the compressor by flexible piping. By this arrangement an extra receiver can be installed should increased storage capacity be necessary.

(xii) Air Inlet Filter

This is a most important aspect of compressor operation, especially air intake compressor working in a dust laden atmosphere. The air intake feed should be drawn from as clean and cool a source as possible and an efficient air filter fitted on the machine. Drawing No. T.9673 shows an air inlet screen. There are many excellent designs of air filters capable of

being used on this job and it would be necessary to have one of the best fitted as it gives protection to the cylinders, pistons and air valves. The filter should be capable of reducing the impurities down to one milligramme per cubic metre entering the compressor, and this should be accomplished by filters having a resistance of between 6-10 millimetres water gauge, such a resistance having little or no effect upon compressor efficiency.





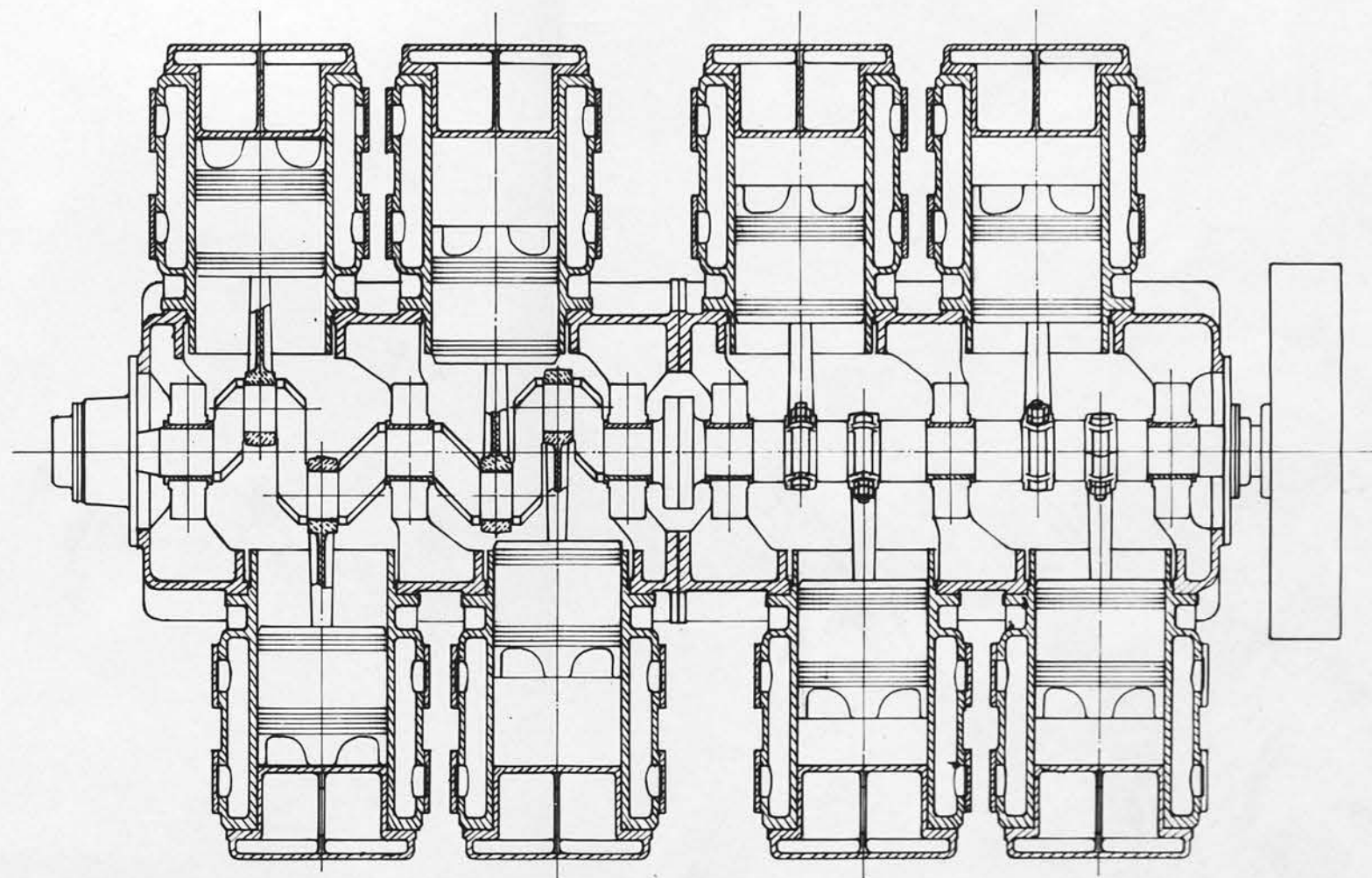
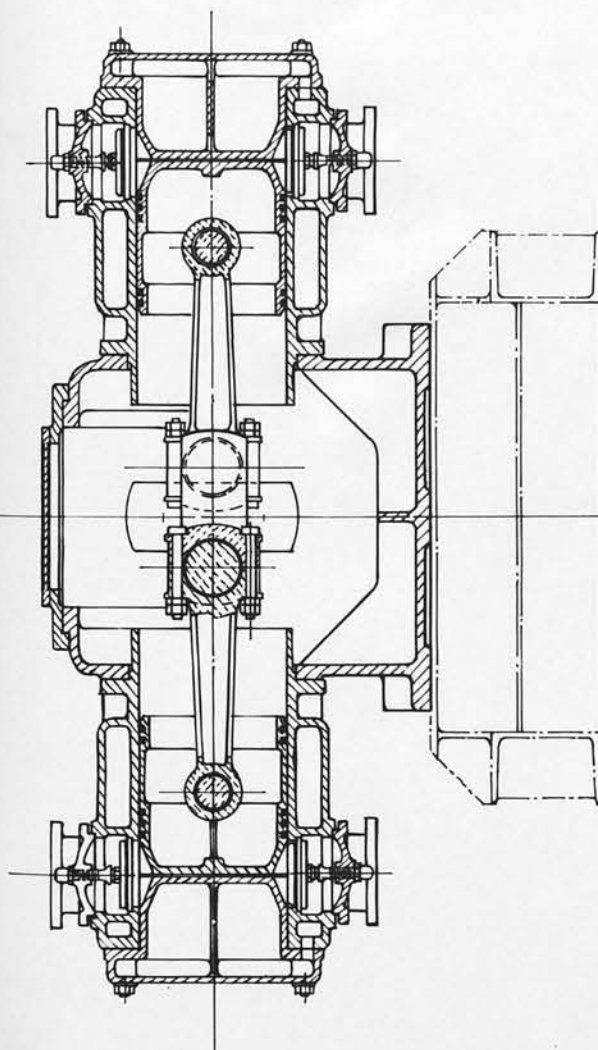
**PROPOSED AIR COMPRESSOR FOR PNEUMATIC STOWAGE.**

CAPACITY :- 3000 CU FT FREE AIR PER MINUTE AT 370 R. P. M.

40 LBS PER SQ INCH GAUGE PRESSURE

SCALE 3/4" = 1'-0"

DRAWING NO. T.9673.



PROPOSED AIR COMPRESSOR FOR PNEUMATIC STORAGE.

CAPACITY:— 3000 CU. FT. FREE AIR PER MINUTE.

40 LBS PER SQ IN. GAUGE PRESSURE.

370 R.P.M.

400 B.H.P

SCALE 1/2" = 1'-0"

(c) GENERAL DESCRIPTION OF PROPOSED COMPRESSOR(i) Overall Dimensions

It was assumed, when deciding the fundamental data, that the inbye compressor would be required to be capable of being installed in a roadway 12' 0" wide by 8' 0" high. This size is a common size of gate road leading to the face and is generally supported by straight-legged arch girders. As the compressor may have to be moved to a new station at frequent intervals, it is of importance as regards cost that there should be no necessity for making an enlarged compressor house. The overall dimensions of the proposed compressor are:-

Length of compressor unit . . .	14' 6"
Max. width of " " . . .	8 10
Max. height of " " . . .	7 0
Length of driving unit . . .	13 6
Max. width of " " . . .	4 9
Max. height " " " . . .	6 0
Length of Receiver Unit . . .	10 6
Diameter of " " . . .	5 0

The height dimensions include the 1' 0" height of the skid fabricated bed plate. The three mountings are of sledge construction fabricated from 12" x 6" channel iron and  $\frac{3}{4}$ " M.S. plate. The overall dimensions of the bed plates are:-

Motor Bed Plate. . . . .	12'6" x 5'6"
Compressor Bed Plate . . .	13'0" x 5'6"
Receiver Bed Plate . . . .	10'0" x 5'3"

The above dimensions give ample clearance in a roadway of the specified size, the built up compressor being shown on Drawing No. T.9673.

(ii) Transportation Dimensions

If the roadway is straight and level, it may



be possible to pull or skid the installation in three separate units to a new position. There is, however, the question of initial transportation and erection and the possibility of transfer either to another district or colliery. It is, therefore, an important factor in the design that the various units should be capable of being dismantled into sections that can be conveniently handled in the confined spaces underground.

In the present design, the heaviest and most bulky part of the equipment is the driving motor. It would be necessary to transport the stator and rotor separately. The stator dimensions, 7'9" long by 5'0" diam., whilst on the large side, are not such as to be unmanageable. The receiver, whilst bulky, should not give any trouble in handling underground.

The compressor unit is capable of being dismantled into small sections. Pipes, valves, pistons, cylinder blocks, crank shafts, etc., can be easily transported, if necessary. The cylinder blocks are approx. 3'0" by 2'0", whilst the crank cases are 5'9" x 3'0".

For limited distances the compressor may be split into two units, each containing a bank of four cylinders, and transported without dismantling. These two units of horizontally-opposed cylinders are bolted together by an oil tight joint, whilst the crank shafts are connected by a spigotted and bolted flange coupling, the overall dimensions being approx. 5'9" long by 8'10"



wide and 3'6" high with the intake pipes removed.

The design, therefore, is such that no serious difficulty is anticipated in installing and transporting the machine underground.

THE ECONOMIC ASPECT OF POWER STOWING

- (a) GENERAL ECONOMIC CONSIDERATIONS.
- (b) LABOUR AND MANPOWER.
- (c) PLANT:     (i) Capital Cost.  
              (ii) Depreciation.
- (d) POWER COSTS.
- (e) OVERALL STOWING COSTS.
- (f) CONCLUSIONS.

THE ECONOMIC ASPECT OF POWER STOWING(a) GENERAL ECONOMIC CONSIDERATIONS

In considering the economics of power stowing, there are many cost items which cannot be assessed, and many savings in other works, whilst not chargeable to power stowing, are the result of the influence of stowing. The primary object is to reduce the cost of working a seam, which may be in the form of an immediate saving in the cost of timbering and packing or in a long term view, of increasing the total workable reserves of coal. The reduction in cost may not be apparent over a short period, but taking the long term benefits of improved roof control, with the consequent saving in roadway repairs, etc., there is no doubt that in the spheres of safety and economy power stowing offers many advantages. The economic advantages of increased safety to workmen, with the probable bearing on labour recruitment and the cumulative benefits to be expected can only be determined on a long term policy. On the other hand, there are conditions, such as the liability of seams to spontaneous combustion or in the working of thick seams especially in undersea areas, where the necessity of economically stowing the goaf is of paramount importance. In some countries, the stowing of the goaf is viewed in the light of total extraction to conserve vital qualities of coal.

The main aspects in this country, excluding the safety factor, are the questions of utilisation

of manpower and the reduction in the costs of production. The economic considerations must be viewed against the background of the rising costs of production in the past few years. In 1938 the national average cost of production per ton of coal commercially disposable was about 16/-. In 1945, it had risen to 35/9d., in 1947 to 41/9d., and in 1948 to 46/-d., whilst in 1949 there was a slight reduction to 45/4d. Obviously these rising costs of production are reflected in the pithead selling price, which in a wider sphere gravely affects the industrial and economical prosperity of the country as a whole. As wages form about 68% of the production costs, any saving in manpower or more efficient use of the available labour force will have the most direct effect on cost reduction.



(b) LABOUR AND MANPOWER

In conventional systems of longwall mining a large proportion, sometimes up to 50%, of the face labour are employed on the non-productive, although essential work of ripping, packing, timbering and withdrawal of timber. Any reduction in the manpower employed in this class of work is a direct saving and is especially beneficial if such men can be utilised in coal getting operations. The mechanisation of this byework is long overdue, but with the introduction of power stowing and hydraulic or other type of roof support considerable saving in manpower is being effected. Whilst it is realised that this class of work will always be necessary, there can be a considerable reduction in the amount of labour expended.

The labour and manpower requirements depend on the amount of stowing to be done per turnover. The amount stowed per shift is dependent upon the proportions of actual packing to total length of face, to the method used at the face and to the capacity of the stowing machine. In Pneumatic stowing 75% of the shift time is spent in actual stowing if stowing solid, but may be as little as 25% with strip packing. The figures should be approximately the same for high speed belt work.

The following table gives a few examples of stowing done in various thicknesses of seams in this country:-

Seam Thickness	Turn- over	Length of face	Strip Packs	Wastes	Gate packs	Stowing Shifts.
5' 6"	4' 6"	140 yds.	7 yds	12 yds	10 yds	2
3' 6"	5' 0"	130 yds.	stowed solid			1
5' 6"	5' 0"	140 yds.	"	"	"	1
7' 0"	5' 0"	110 yds	"	"	"	1

The manshift results with power stowing are best considered by the comparison of figures per 100 tons of coal output, for each class of work in the goaf. These manshifts should include all labour employed, from the preparation of the material, transport underground, in tipping and feeding the machine, in addition to actual stowing and drawing off.

The personnel employed in the transport and tipping of the material depends upon the layout and the efficiency of the system, whether the material is prepared on the surface or underground and to some extent on the actual source of supply of the material.

In pneumatic stowing, teams of 3-5 men, including the chargeman, work on the face, while 4 men set the boxing timber and catch props, draw off and dismantle the pipes. Usually on a non-stowing shift 2 men move up the face bend pipe, extend the gate pipe, clean and oil the machine, etc.

A summary of stowing personnel in three collieries is given for comparison:-

Colliery A (Penallta, S.W.D.)

## Remarks

Steel withdrawing & timber erection. . . . .	3	Stowage shift 260
Assembling pipes and supplying timber. . . .	2	trams at 30 cwts.
Handling stowing pipes etc . . . . .	3	Coal output approx. 500 per shift (meco-Moor.)
Stower Operator . . . . .	1	
Tippler Operator. . . . .	1	
Haulage Hands (including Engine Driver). . . . .	4	
	<u>14</u>	

Colliery B (Ferndale, S.W.D.)

Steel withdrawers . . . . .	2	120 cu. yds.
Pipe Assemblers . . . . .	1	stowing per shift.
Stowing Team . . . . .	6	
Haulage Hands . . . . .	3	Coal Output 250
Pit Bottom Extra. . . . .	1	tons per shift.
Surface Marshalling . . . . .	2	
	<u>15</u>	

Colliery C (Bullcroft, N.E.D.)

Wire bag packer . . . . .	1	Strip Packing
Stowing machine operator. .	1	Two shifts.
Prop Drawers . . . . .	2	Coal off face 290
Face stowing gang . . . . .	2	tons per shift.
Tippler personnel . . . . .	2	
Haulage Personnel . . . . .	2	
	<u>10</u>	

Manshifts per 100 tons coal output

In the Ruhr collieries using pneumatic stowing the average figure is said to be 4 manshifts per 100 tons coal output, although a number of collieries produce figures of less than 3. In the three British cases cited above, it will be seen that colliery A shows 2.8 manshifts per 100 tons and Colliery B is much higher at 6 manshifts. Colliery C with strip packing and total timber extraction, about 7 manshifts per 100 tons coal are required. It can, therefore,

be assumed that a good average figure should be near 4 for solid packing and up to 8 manshifts per 100 tons coal for strip packing. It is obvious that the figures will be reduced if thicker coal is worked, as a stowing team in a thin seam is not reduced in personnel in direct proportion to the thickness of coal worked.

#### Comparison between Hand and Pneumatic Stowing

In normal longwall working it was usual to pay all labour on the face, coal getting, packers, conveyor shifters, etc., on a tannage price. The present practice in power stowing is to pay on a day wage and not proportionate to the amount of coal filled. This complete change in the methods of payment does not allow cost comparison for different types of work, although a comparison can be made on a manpower basis.

Table 2I gives a manpower comparison at a pit in S. Wales between hand packing and power stowing on a face requiring 275 cu. yds, of pack.

TABLE 2I

Hand Stowing	Man Shifts	Pneumatic Stowing	Man Shifts
Tippler . . .	2	Tippler. . . . .	3
Packers . . .	33	Stowing Team . . .	6
Timber Men. .	1	Laying Pipes . . .	$\frac{1}{2}$
Conveyor Att.	1	Boxing Erector . .	1
Charge Hand .	1	Time Study, etc. .	$1\frac{1}{2}$
Fireman . . .	<u>1</u>	Fireman. . . . .	<u><math>1\frac{1}{2}</math></u>
Total	<u>39</u>	Total	<u><math>13\frac{1}{2}</math></u>

The saving shown in the case of the above colliery is, however, not always apparent in other



instances. At Bullcroft Colliery, Yorkshire, the immediate face byework personnel is increased with power stowing, but a manpower saving is effected on back rippings. The total power stowing personnel per turn-over is 23 against the previous face packing personnel of 18, an increase of 5 per shift. This increase is, however, more than recovered, as there is less ripping to be done in the main gate and tailgate, due to the improved roof control.

The balance of shifts on a weekly basis is as follows:-

Pneumatic Stowing Personnel	115	shifts
Back Rippers. . . . .	<u>20</u>	"

Total 135

Hand Packers . . . . .	90	"
Back Rippers . . . . .	<u>50</u>	"

Total 140

Whilst this only shows a saving of 5 shifts in favour of power stowing, the cost difference is more than is apparent in the figures, as the stowers are paid on a day wage, which is less than the old contract system for hand packing.

The saving in manpower and the elimination of arduous work will become more apparent as the technique of stowing improves. Taking an average of 8-12 shifts per 100 tons coal output for handpacking systems, the saving should be in the region of 6 manshifts per 100 tons of coal output, on the introduction of power stowing.

Labour Cost

The Stowing Labour cost per ton of coal varies considerably, depending upon underground conditions and layout and the source and method of material preparation. This cost may vary from 1/- to 3/4d. An average of eleven collieries in South Wales gives 1/- to 1/3<sup>5</sup>d., and it would appear that a suitable average figure for the country to be in the region of 2/6d. per ton of coal mined from power stowing areas. Where provision is made in new collieries for mechanised stowing, this figure should be considerably reduced.

(c) PLANT(i) Capital Cost

The capital cost for plant will vary according to the layout and the ancillary equipment necessary. The total capital cost will depend on numerous factors such as the extent of preparation plant required, whether this is situated on the surface or underground. In some cases little or no preparation will be required. In pneumatic stowing it may be necessary to charge the installation of an air compressor and air main, if this form of power is not in general use underground. The only guiding figures that can be given with any accuracy are the capital costs for plant from the tippler to the stowing machine.

In the case of a straightforward installation the capital cost for plant would be as follows; (approx. 1948):

One 120 cu. yds. per hour stower with Vee engine drive. . . . .	£ 950
One spare rotor and spares. . . . .	200
One Tippler. . . . .	300
200 yds. 6" diam. stowing pipes with patent couplings @ £13.11.0d per 10ft. length .	813
200 yds. 6" diam. flanged stowing pipes for gate @ £8.10.0d per 10ft. length. . . .	510
Two 6" diam. bends with hard inserts. . . .	<u>120</u>
	<u>£2,893</u>

The above table does not include an elevator or feed conveyor to the machine. In another installation the complete stowing plant costs approximately

£3,650 made up as follows, (1949 figures) :-

Tippler (Powell Duffryn) . . . . .	£ 640
Gate and Loader . . . . .	855
Stower (Markham) . . . . .	930
(Beien £916)	
200 yds. road pipes . . . . .	510
150 yds. face pipes . . . . .	650
Bend . . . . .	<u>65</u>

Total £3,650

At Penallta (S.W. Div.) the approx. capital cost for plant is given:-

Stower (Markham . . . . .	950
Feeder. . . . .	750
Tippler . . . . .	600
Road Pipes. . . . .	400
Face Pipes. . . . .	550
Bend. . . . .	<u>65</u>

Total £3,315

#### (ii) Depreciation

The accurate fixing of rates of depreciation can only be determined on some years running and maintenance costs. Different sections of the equipment will depreciate at different rates and even parts of the equipment, although part of one machine, will require separate consideration. This obviously cannot be taken down to items too small, and it is usual to confine depreciation to complete machines, except where an expensive part of the machine wears much more rapidly than other component parts, e.g. the rotor or paddle wheel in a pneumatic stowing machine. In all forms of power stowing the quantity of material which the equipment will handle before renewal depends almost entirely on the abrasive nature of the stowing material. As far as pneumatic stowing



is concerned, it is now possible to give fairly practical figures, based on the averages assessed from widely different conditions and qualities of material handled.

#### Stowing Machine and Parts

Of all the equipment used, this machine suffers the greatest wear and tear. Information is tabulated to show total quantities handled and expected life based on 200 cu.yd. per shift for 250 days work in the year:

Equipment	Cubic yards stowed before replacement	Expected Life
Stowing Machine	150,000	3 years
Rotor. . . . .	30,000	6 mnths.
Shearing Strips.	15,000	3 mnths.
Side Jaws. . . .	100,000	2 years.
Discharge Jets .	100,000	2 years.

#### Pipe Lines

Pipe line life may be considerably extended by careful inspection and maintenance, as the greatest wear takes place on the bottom of the pipe, which should be turned at suitable intervals. If special type joints are used, these tend to get damaged by frequent making and breaking of joints. Face pipes at the top of the face and the bends suffer most wear. Average figures show that bends can be expected to pass 15,000 cu. yds. and have a life of 3 months, whilst the pipe lines should convey 125,000 cu.yds.

and have a life of approximately  $2\frac{1}{2}$  years. Road pipes are generally rated with a longer life, and are depreciated at 20% per annum.

#### Feeding Equipment

This consists of colliery machinery, and is usually depreciated at normal rates, i.e.

Feeder Conveyor. . . . 20% p.a.

Tippler. . . . . 20% p.a.

The actual depreciation and overhaul cost per ton of coal varies from 1.6d to 3.5d. In estimating costs under this heading, allowing for the use of fairly hard material, 3d. per ton should be regarded as a suitable figure.

(d)

(d) POWER COSTS

Considering the three systems of stowing, pneumatic stowing is the most expensive in power consumption. On a horse-power basis the ratio is about 7 to 1 in favour of mechanical stowing: (60 H.P. as against 350-450 H.P.). Reliable figures are not available for hydraulic and mechanical, but pneumatic costs have been closely investigated. To obtain the most economic air consumption it is necessary to run the equipment at the full capacity with the lowest practical air pressure, to obtain the lowest possible ratio of cubic feet of air per cubic foot of material stowed. With the present design of machine, the lowest cost per cubic foot stowed is obtained with high capacity machines as shown in the following table:-

Machine Capacity	Cu. ft. air required per min.	Cu. ft. air per cu. ft. stowed.	Cost per cu. ft. stowed.
35	1,600	100	0.25d.
55	2,000	80	0.20d.
120	3,000	56	0.14d.

Compressed air cost is usually reckoned at 2d-2½d per 1,000 cu. ft. The total power cost per ton of output varies from 3d to 7½d. The majority of the installations running in this country have a power cost in the region of 3d-4d. per ton. Although the consumption of air is high, if the supply of material is good and the stowing machine efficiently run by the operator, the cost is not out of proportion with other

costs in power stowing.

(e) OVERALL STOWING COSTS

The total cost of power stowing and of any projected scheme of stowing is usually a subject of considerable controversy. Any set of cost sheets can be misleading unless the items included are exactly specified. Total costs of this form of stowing should, therefore, include all costs applicable to the work done, whether wholly or partly employed. The headings can be conveniently grouped under labour, depreciation of plant and equipment, power, transport and haulage, preparation and supply of stowing material.

The following Table, abstracted from results obtained in S. Wales, gives the itemised costs of 11 collieries using pneumatic stowing \* calculated on the cost per ton of coal (gross).

Coll- iery	Labour cost		Stores cost		Power cost		Depreciation cost		Total cost	
	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.
A	1	8.1	0	1.6	0	3.0	0	1.6	2	2.3
B	2	0.2	0	0.7	0	4.1	0	1.9	2	6.9
C	1	4.0	1	4.3	0	3.7	0	1.6	3	1.6
D	1	11.3	0	11.2	0	4.5	0	1.6	3	4.6
E	1	9.4	0	3.4	0	5.2	0	1.6	2	7.6
F	0	9.9	1	2.5	0	6.8	0	2.0	2	9.2
G	0	11.6	1	6.8	0	6.2	0	1.8	3	2.4
H	0	9.0	1	10.3	0	4.5	0	1.9	3	1.7
I	1	3.4	1	1.5	0	4.2	0	1.6	2	10.7
J	0	10.9	1	1.2	0	3.5	0	1.9	2	5.5
K	0	10.0	0	8.8	0	4.6	0	3.5	2	2.9
Avg	1	3.5	0	11.3	0	4.6	0	1.9	2	9.3

The above figures relate to complete solid stowing in varying conditions and systems, but using

\* Pneumatic Stowing, H.F. Landsdowne, M.Sc.,  
Coll. Guard., Vol. 178, 24.2.'49.



approximately the same type of face equipment. The average total cost of 2/9d. can, therefore, be taken as a good estimating figure.

In comparing total costs between systems of solid stowing and strip packing, the following table, from the same source, gives an interesting comparison. At the Yorkshire colliery, the packing is high quality strip packing and the total cost is approximately 2/7d. per ton, as against the S. Wales average of 2/9d. for solid stowing:-

Cost Factors	Percentage of total cost	
	S. Wales	Yorks.
Labour: Preparation, transport, stowing team, prop withdrawers. . . . .	46	69
Stores: Stowing support and rimber. . . . .	34	0
Power: Compressed Air at 2½ per 1,000 cu. ft. . . . .	14	15
Depreciation: Tippler Feeder, stowing machine and pipe lines. . . . .	<u>6</u>	<u>16</u>
	<u>100</u>	<u>100</u>

The stowing cost per ton of coal output varies considerably not only with the layout of the stowing installation, but also with the thickness of seam worked and the method of working. The following cost notes are abstracted from figures given in a Report by the Stowing Committee of the N.C.B. \*

6 6

Colliery A (Scotland). Room and Pillar working in a 20 ft. seam with solid stowing. Total cost per ton of coal is 2/10d to 4/3d., including 4d-7½d. for power and 3½d for interest and depreciation on surface preparation of the material.

Colliery B (Scotland). Longwall working in a 28ft. seam. Cost 4/6d per ton of coal, including 7d. for power, 4d. for depreciation and 1/5d. for underground crushing.

Colliery C (Yorkshire). Stowing gate side packs on all units and strip packs on three double units. Packs 7 yds. wide and wastes 12 yds. Labour cost 1/9d. Power and Depreciation 9½d. Total cost 2/6½d.

Colliery D (South Wales). Three Districts stowed solid in a 4 ft. seam. Cost of best unit is 3/9½d. with 1½d. for surface labour 3½d for power, 1/8½d. for stores and 1½d. for depreciation.

Colliery E (South Wales). Solid stowing in a 3' 2" seam. Total cost 2/2d., including 3d. for power, 1.6d. for stores and 1½d. for depreciation.

It would appear that the overall cost of stowing operations based on present day prices varies from about 2/3d. to 4/6d. per ton of coal mined from the stowed areas. Analysing the above figures and taking into account the varying methods of work and thickness of seam, it is seen that thick seams are the most expensive conditions in which to introduce mechanised stowing. The two cases cited, however, have certain special factors to be considered, which are unusual and do not consist of normal conditions, i.e. the liability of the seam to spontaneous combustion and undersea mining operations. For seams of average thickness, working longwall, the total cost varies between 2/3d. and 2/6d.

(f) CONCLUSIONS

As discussed in the opening paragraphs and under subsection (b), direct saving in manpower is desirable and possible even in conventional systems, and will become more so in the future when collieries are projected with ample arrangements for mechanised stowing. Considering the capital costs involved and the saving in manpower achieved, excluding cumulative benefits, the amounts are well within the theoretical warranted capital expenditure limits required to replace manpower. Assuming the average annual wage of the labour employed on this work to be £300 per person at 3% this represents a remunerative capital outlay of approx. £11,670. This capital expenditure, however, is greatly reduced when depreciation, operational and maintenance costs are considered. In the case of stowing operations, the capital warranted to replace manpower can be approximately reckoned:-

C = Maximum warranted capital (£)  
 n = Average life of plant = 3 years  
 R = Ratio of men displaced x annual operational costs.

W = Annual wages per person (£) = 300

Interest on capital taken at 3%, then

$$\begin{aligned}
 W &= \frac{3}{100} C + \frac{1}{n} C + R \\
 &= \frac{3C}{100} + \frac{C}{n} + R \\
 W - R &= \frac{3Cn + 100}{100n} \\
 \frac{W - R}{C} &= \frac{3n + 100}{100n} \\
 (3n + 100)C &= 100n (W - R) \\
 \therefore C &= \frac{100n (W - R)}{3n + 100} = \frac{300(W - R)}{109}
 \end{aligned}$$

Taking the annual wage of a worker employed in this work to average £300, then the formula becomes

$$\text{Capital expenditure warranted} = C = \frac{300(300 - R)}{109}$$

$$\text{when } R = \frac{1}{\text{Reduction in personnel}} \times \text{Annual cost of power and lubrication.}$$

By substituting actual figures from cost sheets, a very low warranted capital expenditure to replace one man is shown. Taking an example:-

#### Colliery A

Power cost = 4.66d. per ton.

Output per day = 290 tons

Average saving in manpower 10

Lubrication costs, say, £50

$$\text{then } R = \frac{1}{10} \times \frac{4.66 \times 290 \times 250}{240} + 50$$

$$= \frac{1440}{10} = 144$$

$$\text{hence } C = \frac{300(300 - R)}{109} = \frac{300(300 - 144)}{109}$$

$$= \underline{\underline{£430}}$$

#### Colliery B

Power cost - 3.0d. per ton

Output per day - 250 tons

Manpower reduction - 10

Lubrication costs - £50

$$\text{then } R = \frac{1}{10} \times \frac{3 \times 250 \times 250}{240} + 50$$

$$= 83$$

$$\text{Hence } C = \frac{300(300 - 83)}{109}$$

$$= \underline{\underline{£600}}$$



An average figure for warranted capital to replace one man in normal industrial undertakings would be in the region of £3,000 - £4,000. It would appear that, considering underground layouts involving plant which has a high rate of depreciation, the capital warranted for manpower reduction is very low. Taking an amount of £1,000 to cover more unsuitable conditions, it still offers an attractive proposition for the installation of this form of mechanisation underground.

The above assumption, however, does not take into account the preparation and supply of stowing material, which with a surface plant would increase the warranted capital necessary. The capital expenditure required for a preparation plant would depend upon local conditions and the extent of preparation necessary, and in layouts investigated varies from £6,000 to £16,000. Underground crushing and preparation may involve even higher capital charges. There is, however, no doubt that in the majority of cases of planned layouts the warranted warranted capital expenditure would be well within the normal accepted limits.

PRESENT APPLICATION AND FUTURE TRENDS IN POWER  
STOWING.

(a) PRESENT APPLICATION

(b) FUTURE TRENDS

PRESENT APPLICATION AND FUTURE TRENDS IN POWER STOWING(a) PRESENT APPLICATION

Except in countries where local conditions make its use economical, hydraulic stowing has been superseded in Europe and in this country by the pneumatic method. In Poland and Germany, hydraulic stowing was used extensively in their special conditions, but except for a few installations in Scotland the system was not generally applicable to British conditions. At the present time hydraulic stowing is extensively practised in Indian coal mines and its use is likely to be extended. Similarly, in metalliferous mining, hydraulic flushing of mill taillings is being increasingly practised. In Great Britain this system has now been completely abandoned.

The pneumatic method is at present the most extensively used system of power stowing both on the Continent and in this country. In Germany, approx. 13% of the output of seams inclined up to 25 degrees in the Ruhr coalfield is obtained from pneumatically stowed districts. The faces are usually solid stowed, strip packing not being extensively used, except in so far as gate packs in roof caving methods.

In Dutch mining practice, the use of pneumatic stowing is becoming increasingly popular, especially for the protection of stone roads on the horizon above the face being worked, and also for solid stowing in thick seams, thereby controlling and minimising surface

damage. The general layout of horizon mines simplifies the economical introduction of this form of power stowing.

The application of powerstowing in this country is in its infancy, but in the last few years has been increasingly developed in three British coalfields. The pneumatic system is easily adaptable to our present conventional working methods, and at present there are some 40 machines at work in the three areas.

In the Scottish Division, the Wemyss Coal Co., were the first to pioneer pneumatic stowing in this country with the introduction in 1937 at Lochhead of a Beien type machine. Hydraulic stowing had previously been used and had become expensive, due to long pipe lines. The Dysart Main (20ft.), a seam liable to spontaneous combustion, was successfully extracted by room and pillar, the stowing enabling the safe and economic extraction of the pillars. The company later extended the system to Michael Colliery, where the same seam was worked longwall in four lifts.

In Yorkshire, the Doncaster Amalgated Collieries introduced pneumatic stowing at Bullcroft Colliery in 1944, where special high quality incombustible packing was required in the Barnsley seam to prevent spontaneous combustion. The advantages gained encouraged the company to extend its use from gate side packs to face strip packs throughout the Barnsley seam at this colliery and this system has later been introduced in several collieries working the same seam in this area.

The most recent extensive introduction of



pneumatic stowing has taken place in South Wales. In 1945 Powell-Duffryn installed pneumatic stowing at Penallta Colliery where solid stowing was necessary on longwall faces due to weak roof conditions and heavy strata pressure. These conditions are common in this coalfield and the success led to the rapid introduction in other collieries. At present there are 18 installations working in 13 collieries and many more projects are contemplated. At present the pneumatically stowed faces produce some 5% of the Divisional output.

The use of pneumatic stowing is gradually being extended in all Divisions, some 26 projects already being in course of installation in Yorkshire, S. Wales, East Midlands and North Western Divisions, whilst other divisions are actively contemplating its introduction.

Mechanical stowing machines are in use in Germany, although employed only in one small area of flat seams, some 15 or so machines being in use. In this country the development of high velocity machines for putting on a tight pack has got beyond the experimental stage and several machines are ready for production work. There are projects to install these machines in the North Eastern and West Midlands Divisions. Two of these installations are proposed in pillar workings and one to work over the face belt on a retreating face.

(b) FUTURE TRENDS

The future development of power stowing must be considered against the background of modern trends in mining layout underground. The possible extension of horizon mining will facilitate and encourage power stowing, especially mechanical stowing, where the retarding factor has been the efficient supply of material to the machine. There is no doubt that the general conclusions reached in the past regarding hydraulic stowing remain substantially true today, but with the experience gained in recent years on stowing methods and the general improvement in pump design and capacities, there will be circumstances where hydraulic stowing may be practised with advantage.

The immediate trend is likely to be towards low pressure pneumatic stowing, as this system is sufficiently flexible for thick seams lying on gradients and for thin seams. Two aggravating factors in future mining practice, viz., the dirt output consequent upon coal face mechanisation in poor quality seams and the possible legislation controlling surface tipping, will stimulate the search for an economic solution of dirt disposal underground. For face stowing it would seem from an examination of all factors, that the trend will be towards pneumatic stowing with an improved machine, constructed in wear-resistant alloys and powered by a low pressure high capacity inbye compressor.

Mechanical developments required for the

introduction of continuous mining include the ripping and packing of road heads, and intensive work will be concentrated on the design of a rippings stower. The successful design of such a machine would have a wide field of application. The growing tendency is likely to be towards gobbing the dirt as produced both on the face and at the road head, together with the transport of washery and other refuse, from the surface to underground.

Facilities for the preparation and supply of stowing material will have to be considered as of equal importance to the getting and transport of the coal, and the layout of a new mine will be influenced by these factors. The indication is that power stowing will increase considerably in the future and will not be considered as a necessary evil due to some adverse condition, but as a necessary operation in the successful mechanisation of coal mining.

## CONCLUSIONS

After a comprehensive study of the possibilities of mechanised stowing, the following conclusions are based on the information resulting from the investigation.

### Development in Mining Technique

(i) The continued development in the technique of mining coal intensively by power loaders is dependent upon the successful mechanisation of the packing and timbering operations. There is doubtless an extensive field for the application of mechanisation to this work to keep pace with continuous mining.

### The Effect of Coal Extraction on Strata

(ii) From the economic point of view, considering the annual subsidence damage figures, the possibilities and advantages of modern mechanised mining are tremendous and fully warrant the closest attention of responsible mining engineers. The question resolves itself into a matter of urgency to minimise the effect of subsidence by exploring every avenue of modern mining technique.

### Preamble on the Necessity for Mechanised Stowing

(iii) Of all the operations in mining, goaf stowing has resisted mechanisation the longest. The requirements necessary are that any method of stowing should be capable of dealing with large quantities of stowing material economically and thus keep back work



in pace with the mechanised getting and handling of coal from the face

(iv) In the past it has been usually considered that the mechanisation of packing, especially solid packing, was costly to instal and operate. Under present day conditions the ordinary system of hand packing and with drawing goaves is becoming a more and more costly operation. The far reaching benefits that accrue from efficient mechanised stowing leave no doubts as to its justification in new layouts, and with future trends in power loading, is probably an essential in most seams to allow the fullest success of the continuous system of mining. The future of continuous mining is of such importance that every effort must be made to produce suitable conditions in which the system can be effectively worked.

(v) Complete extraction of our remaining coal resources is essential. There are large areas of valuable coal standing in pillars to afford support, and large resources lying under built-up regions which in the national interest should be worked. It is possible that this coal can be worked with a minimum amount of surface damage, provided sufficient thought is given to the planning of workings and the methods of packing

#### Mining Layout for Power Stowing

(vi) The horizon system of mining possesses many advantages over the conventional British practice,

both as regards supply and handling of stowing materials and the transport of coal. Where extensive power stowing operations are contemplated in a new project, the provision of facilities as in the horizon system is necessary for success.

(vii) The considerable age of many British mines and the long distances from the shaft to the working face increase the difficulties in altering the layout, and from an economical aspect may be so formidable as to preclude the general large-scale introduction of power stowing in many cases.

(viii) The three systems of power stowing<sup>p</sup> packing material into the goaf can be applied to any system of mining, with the proviso in the case of hydraulic stowing that the seam conditions as far as gradients are concerned are suitable. Examples of underground layouts are given, with the three systems, to illustrate the applicability to different conditions.

#### Stowing Materials

(ix) The nature of the stowing material to be used is of paramount importance. The amount of support given by the packs depends directly on the compressibility of the material, together with the completeness of the packing. In this country it would appear from enquiry into present materials used, that pit rubbish, washery debris, or crushed rock from underground rippings, can be stowed successfully, either mixed or used separately.

(x) In general, coal measure stratum provides a suitable stowing material, admixtures varying for each type of shale or rock determined for correct proportion by practical test.

(xi) From the tests described, it can be deduced that a wide range of sized material can be successfully stowed. The limit of sizes varies with the system used, and as far as hydraulic and pneumatic methods are concerned is closely linked with the pipe diameter. It can be generally accepted that the pipe diameter should be twice the maximum round material size.

(xii) In pneumatic stowing a maximum size of 3" is suitable for general application. In hydraulic stowing the percentage of scouring size can be much lower, that is, small sized material can be successfully handled without considering the larger sizes as is necessary in pneumatic stowing. In mechanical stowing the limiting size appears to be about 4" with the present machines, but if necessary this can be increased with little difficulty.

(xiii) To give a good pack density, a stowing material should contain not more than 25 % of minus  $\frac{1}{8}$ " fraction if of a hard nature, and not more than 15%, if of a friable nature. This with 50% above 1" size would give an efficient packing medium.

(xiv) The shape of the pieces of stowing material not only governs the way they fit into the pack and thus have an important bearing on the load

distribution, but also the shape has an important bearing on the stowability of the material.

(xv) Taking the ideal ratio as unity proportions, a suitable material should not have a ratio of length to breadth of more than 1-3 and thickness to breadth of more than 1 to 5.

(xvi) The main practical factor involving shape is the effect of shape on the surface area, as the efficiency with which a particle will be conveyed along a pipe either by compressed air or water is governed mainly by the density and surface area.

(xvii) The shape factor plays an important part in the theory of air/water conveying and it is shown that this factor (K) can be calculated from the Formulae:-

$$\text{Surface Area} = \frac{6K}{pd}$$

from which it is deducted that K

$$= 0.78S \text{ for sandstone}$$

$$= 0.703S \text{ for shale.}$$

(xviii) For sandstone the shape constant varies between 1.0 and 1.5, an average of ten examples giving 1.2, whilst for shale the constant lies between 1.0 and 1.68, the average for ten examples giving 1.18. For average material using the above constants the surface area for a 1" -  $\frac{1}{2}$ " fraction was reasonably correct.

(xix) Using the above shape constant values, the surface area in sq. centimetres per gramme:



$S = 7.2/pd$  for sandstone

$= 7.08/pd$  for shale

where  $d$  = mean size in cms.

$p$  = specific gravity of the material

These formulae give a convenient method for estimating the surface area of material lying between sieve sizes when the specific gravity is known, and thus the shape of the material can be assessed in terms of the surface area for consideration of the suitability for water or air conveying.

(xx) The density or compactness of a pack depends upon several factors, but the combined effect of these factors can be expressed in the percentage of voids contained in any pack. From tests, the void percentage in normal stowing material lies between 40-48% whilst tests on various sized materials showed that there is no direct connection between the size of the material and the percentage of voids. Even two materials which come within the limits of the same sieves may contain different percentages of voids.

(xxi) Together with particle shape, the amount of small material in the general body of the material has an important effect on the void percentage. The addition of small material to give minimum voids varies according to the type and size of packing. From the tests it would appear that stowing material of close sizing is not as suitable for underground packs as suitably sized material.

(xxii) The degradation of material whilst in

transit may, therefore, have a beneficial effect on the resistance characteristics of the pack by causing a reduction in the void percentage. These tests show that theoretically more efficient support should result in the use of small sized material for pack building.

(xxiii) One important feature disclosed in these tests is the high percentage of voids to be found in the materials used for underground packing and stowing. With mechanised stowing it is essential to develop means to pack the material tight either by increasing the speed of propulsion into the goaf or by some ramming arrangement.

(xxiv) The present system of hand packing in the majority of cases cannot be efficient, as the resistance by the pack is offered at a late stage in the operations. In view of the doubtful value of hand packing as a means of roof control and the high cost involved, it is of importance that this phase of underground mining should be critically examined.

(xxv) The amount of support given by the packs depends directly on the compressibility of the material used together with other factors. The need for a standard test to evaluate the suitability of material available for the construction of packs is essential in view of the increased use and probable large-scale development of mechanised stowing.

(xxvi) The results obtained by testing packing material in a barrel and plunger apparatus can give an indication of the ultimate behaviour of the material

under compression. If the initial compression or flow of a pack is determined, the final maximum compression can be deduced.

(xxvii) The compression tests show in general that in all sizes fractions the major part of the compression occurs before one third of the load has been applied. The compression rate is greatest with the initial loads and it appears that there is little effective resistance to compression until the material has been reduced to approximately 60% of its original volume.

(xxviii) For shale type materials the initial compression increases with size reduction, whereas for sandstone type materials the percentage compression decreases with decrease in size.

(xxix) Sand is shown to have little compression and that compression is spread evenly over the load. This may be due to interparticle binding, a characteristic not apparently found in other material. Theoretically, sand is, therefore, an eminently suitable admixture.

(xxx) Deduction from the tests shows that the height of a pack does not affect the percentage compression during the initial pressures, but the divergence increases as the load is increased. The stage of effective resistance to compression is reached at a lower load in a thin pack than with a pack of greater height.

(xxxi) The final percentage compression of a

sized material approaches closely to the figure of the original void percentage of the material. The larger fraction of the sandstone samples showed the highest final compression, due to a greater degree of point contact, than the corresponding size in shale. The smaller sized sandstone fractions, however, exhibit a greater resistance than their shale counterpart both at initial and final loads.

(xxxii) At some stage in compression, factors other than void content play an important part in the resistance to pressure. The crushing strength of the material, parallel and at right angles to the stratification, will be the criterion at high pressures, whilst the liability to disintegrate and thus compact the interstices will help to resist compression at lower pressures.

#### Material Handling

(xxxiii) The problem of efficient preparation, transport and feed arrangements, are of vital importance for mechanised stowing, as they determine whether or not any system can be successfully applied to any given set of conditions.

#### Stowing Pipes

(xxxiv) Tests and practical experience indicate that the hardened steel pipes, with a wear-resistant lining of hardened steel of high ultimate tensile strength, is superior to softer types of steel where frictional abrasive forces are predominant.



(xxxv) The lining of stowing pipes with basalt or other wear-resisting material is undoubtedly advantageous, but further investigation, especially into manufacturing problems, is required.

(xxxvi) In certain specific instances, especially in hydraulic stowing, there is much to said for suitably designed cast iron pipes. Fine grained chilled cast iron pipes for stationary locations may show favourable economic balance as compared with drawn steel.

#### Survey of Stowing Plant

(xxxvii) In coal mining, hydraulic stowing has many disadvantages, amongst which are the effect of water on soft strata and the added burden of pumping from great depths. Where conditions permit its use, however, there is much to commend the system, and its popularity is increasing in many parts of the world, more especially where ample supplies of sand are available near the mining areas.

(xxxviii) Pneumatic stowing systems can be used in all thicknesses of seams on any gradient, and can be introduced with a minimum of inconvenience at the face, to suit all existing working methods. The main disadvantages of pneumatic stowing are the high consumption of air and the difficulties of supplying the air to feed the system.

(xxxix) To provide an efficient and economic supply of compressed air, it is desirable that the compressor should be an inbye machine of suitable

capacity, so constructed as to be easily dismantled and erected in the confined spaces near the coal face. Details and general outlines are given of such a machine, specially designed for inbye compression, having a capacity of 3,000 cu.ft. of free air per minute at 40 lbs. per square inch pressure.

(xl) Mechanical stowing in any form is not flexible, the material having to be transported and fed into the machine at the stowing site. The machine has also to be handled in the confined spaces of the coal face. However, the importance of its ability to handle a great variety of stowing material, and the considerably less consumption of power, make the successful application of this method very desirable.

(xli) The development of a suitable high speed belt machine is necessary and theoretical details are given of such a machine.

#### The Economic Aspect of Power Stowing

(xlii) With power stowing, the general reduction in costs may not be apparent over a short period, but taking the long-term benefits which accrue, there is no doubt that in the spheres of safety and economy, its introduction offers many advantages.

(xliii) Amongst the vital problems confronting the mining engineer today, excluding the safety factor, are the questions of utilisation of manpower and the reduction in costs of production. As wages form about 68% of the production costs, any saving in

manpower or more efficient use of the labour force will have a direct effect on cost of production. Taking an average of 8-12 manshifts per 100 tons coal output for hand packing systems, there would be a considerable saving on the introduction of power stowing.

(xliv) Power Stowing plant suffers heavy wear and tear, the actual depreciation costs per ton of coal varying from 1.6d. to 3.5d. In estimating depreciation costs, 3d. per ton of coal should be regarded as a suitable figure, allowing for the use of fairly hard material.

(xlv) Considering the three systems of stowing pneumatic stowing is the most expensive in power consumption. On a horse-power basis, the ratio is about 7 to 1 in favour of mechanical stowing. The majority of installations running in this country have a power cost in the region of 3d.-4d. per ton. Although the air consumption is high, if the stowing machine is efficiently run by the operator, the cost is not out of proportion with other costs in power stowing.

(xlvi) The overall stowing cost per ton of coal output varies considerably, not only with the layout of the stowing installation, but also with the thickness of seam worked and the method of working. It would appear that this cost based on present-day prices varies from about 2/3. to 4/6d. per ton of coal mined from the stowed areas. For seams of average thickness, working longwall, the total

cost varies between 2/3d. and 3/6d.

(xlvi) In underground layouts involving stowing plant which has a high rate of depreciation, the warranted capital expenditure for manpower reduction is well within the normally accepted limits.

#### Future Trends in Power Stowing

(xlvi) The immediate trend is likely to be towards low pressure pneumatic stowing, as this system will have the most general application in British coalfields.

(xlix) The introduction of legislation controlling the surface tipping of colliery refuse will stimulate the search for an economic solution of dirt disposal underground.

(1) In the layout of new mines, power stowing in all its phases will have to be considered as of equal importance to the getting and the transport of the coal. Power stowing will increase considerably in the future as it will not be considered as a necessary evil due to some adverse conditions, but as an essential operation in the successful mechanisation of coal mining.